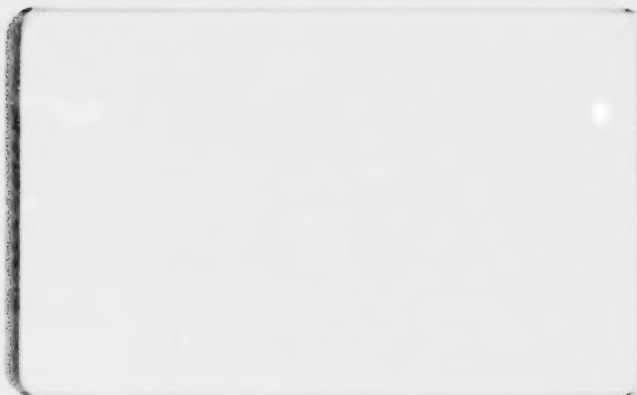




canada/yukon subsidiary
agreement on mineral resources



Canada

Yukon
Government

Program 3 Placer Mining

FINE GOLD RECOVERY OF SELECTED
SLUICEBOX CONFIGURATIONS

University of British Columbia



DEPARTMENT
OF
MINING
AND
MINERAL
PROCESS
ENGINEERING

PILOT SCALE STUDY OF:

"FINE GOLD RECOVERY OF SELECTED SLUICEBOX CONFIGURATIONS"

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Executive Summary

1. Objectives

The Government of Canada and the Yukon Territorial Government, through the Economic Development Agreement for the Yukon Territory, jointly funded a study of the fine gold recovery of selected sluicebox configurations. The major objectives of this study were:

1. To examine how variations in sluice operating conditions affected recovery of gold from different size fractions (down to 150 mesh or 105 microns) during sluicing operations.
2. Based on the results of the study, provide recommended operating conditions that yield high overall gold recoveries and low water use in sluicing operations.

2. Conclusions

1. Sluiceboxes, operating at waterflow rates and solid feed rates typical of the Yukon placer mining industry (from 100-700 lb solids/min/ft sluice width) can be an effective method of recovering gold as fine as 150 mesh or 105 microns.
2. Expanded metal riffles (such as the 1-10H) are superior to 1 1/4" dredge riffles for recovering placer gold between 20 and 100 mesh.
3. The orientation of expanded metal riffles is not important to gold recovery.
4. The practice of running "clean" or allowing the gravel feed to stop while water is flowing need not greatly affect recovery.

5. Contrary to other studies of sluicing, (1), our results indicate that coarsening the upper size of gravel in the feed from 1/4" to 3/4" does not significantly influence recovery.
6. The scour condition that exists in the sluice is the most significant factor in predicting recovery. Each riffle type has a characteristic scour condition where gold recovery is optimal.
7. Normal variations in the solid feed rate can be tolerated in sluicing without excessive gold losses.
8. Recoveries in excess of 95% down to 100 mesh are possible using process water with a high suspended solids content (> 10,000 ppm).
9. Low water use is beneficial to gold recovery. The best recoveries using expanded metal riffles were obtained using a water to solid ratio by weight of approximately 4:1.
10. Nomad matting and Cocoa matting are both effective at retaining gold when exposed during scouring. Nomad matting is much easier to cleanup.

3. Recommendations

1. Placer miners should use expanded metal as the sluice riffle of choice for fine gold recovery from feeds of -1" placer gravel.
2. Solid feed rates up to 700 lb/min/ft sluice width are acceptable over 1-10H expanded metal under suitable scour conditions. At this feed rate, recovery of fine gold (-65+100#) could be slightly less than 90% but overall recovery of the gold sample including the coarser gold fractions, can exceed 90%. The recommended operating procedure is to use 300-400 lb/min. of solids per foot of sluice width accompanied by approximately 200 USGPM water at a slope of 1 5/8 - 2"/ft. Gold recoveries as high as 95% of the -65+100# gold should be achievable. Higher slopes, attendant with some sluice designs, would require less water flow

but would be more sensitive to fluctuations in solid feed rate. However, lower solid feed rates, under suitable conditions, give marginally better recoveries.

3. Angle iron dredge riffles should be used somewhere in the fine gold recovery area to recover gold particles much coarser than 20 mesh and smaller than the upper feed size. Frequently referred to as a "nugget trap" when used in this manner, the dredge riffles would serve to capture gold particles too large to be retained in the relatively low profile expanded metal riffles. Except for extremely flat particles, which might be caught in the fine gold riffles, the recovery of the +10 mesh gold nuggets should be high in dredge riffles. An ideal location for such a "nugget trap" would be at the discharge end of the sluice. This location would allow the fine gold riffles to process the well sorted slurry at the sluice entrance. In this manner the maximum amount of fine gold could be recovered in the more efficient riffles prior to passing over the less efficient, more turbulent, dredge riffles. The gradient of the "nugget trap" portion of the sluice could also readily be changed to produce the appropriate scour without influencing conditions in the fine gold riffles.
4. Many valuable data could be gathered by obtaining details of cleanup results from selected, cooperative sluicing operations. Cleaning the fine gold recovery areas in sections according to distance from the feed and, sizing the recovered gold would involve considerable extra time and effort but the data generated could prove beneficial to the entire placer mining industry.
5. Placer miners should investigate the effects of having short lengths of smooth, unriffled sluicebox base in their fine gold recovery sections. An example would be to have 4' of riffles at the feed end of the sluice followed by 2' of smooth base (no matting). By alternating sections of riffle and smooth base the slurry entering each riffle section would be pre-segregated so that a proportion of the high density minerals would be flowing along the base of the flow. This might counteract the tendency of

the recovery probability to decrease as distance from the feed end of the sluice increases.

6. Future research on sluicing would be beneficial if directed towards investigating the recovery of gold when:
 - a) using different riffle types
 - b) processing different gravel types, with or without bedrock fragments
 - c) using very fine gold (-150 mesh).

4. Description

A pilot-scale, sluice facility was constructed to carry out the test program. The test facility consisted of a 12 inch wide by 8 ft long sluice that received a slurry of gold bearing test gravel and process water from an entry flume. Gravel was introduced to the entry flume from a 2 cubic yard feed hopper fitted with a well sealed belt feeder. Control of solids feed rate was achieved using a variable speed motor on the belt feeder. A closed circuit system was used for the process water to prevent losses of clays and other fine solids.

The gravel used for the test program was obtained from screening 15 tons of gravel from Teck Corporation's Sulphur Creek placer operation in the Yukon. The gold seeded into this gravel was obtained from this same placer mining operation. Three size fractions of gold (-20 +28 Tyler mesh, -35+48 Tyler mesh, and -65+100 Tyler mesh) were used for 24 test runs. One test was conducted using -100+150 Tyler mesh gold.

The -1/4" fraction of the gravel sample was used for 19 test runs and then the -3/4" fraction was added to this feed gravel during 6 test runs.

The gold bearing concentrate that remained in the 8 ft. long test sluice, upon completion of each test run, was removed in 2 foot sections. The gold of each size fraction was extracted, dried, and weighed. The entire gravel sample down to fine silt sized particles was recovered at the end of each test. The gold and gravel were remixed in batches for reuse in subsequent tests.

This research was funded by the Government of Canada through the Department of Indian Affairs and Northern Development in Whitehorse via a contract with the Yukon Chamber of Mines. Support of this project from the Klondike Placer Miners Association and Teck Corporation is also much appreciated!

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A STUDY OF THE FINE GOLD

RECOVERY OF SELECTED SLUICEBOX CONFIGURATIONS

1. Introduction

A sluicebox consists of one or more flumes through which a slurry of water and alluvial gravel is passed. These flumes are rectangular in cross-section and lined along their base with devices called riffles. Turbulent eddies are formed in the slurry as it flows over and around the flow obstructions that comprise the riffles. The interaction of these eddies with the particulate material that tends to collect around the riffles forms a dispersed shearing particle bed where particles of a high specific gravity are concentrated.

Man has used sluicing for the concentration of high density minerals, especially gold, for centuries. Sluicing has remained the preferred mineral processing technique for the treatment of placer gold bearing alluvium. In the Yukon today, the sluicebox is by far the predominant primary concentrator in use for placer gold mining. There are almost as many variations in design or use as there are sluices. This indicates that, although the sluicebox is perceived to be the most cost effective device, there is considerable difference of opinion as to how it should be configured and operated.

It is generally agreed by the operators of Yukon placer mining operations that sluice-recovery of placer gold particles finer than 200 mesh (74 μm) is relatively low. The particle size where recovery starts to decrease significantly is not agreed upon, however. Some sources, particularly the vendors of non-sluicing equipment, maintain that sluice efficiency drops when gold particles smaller than 20 to 65 Tyler mesh (600 μm to 212 μm), depending on the source, are processed. Conversely other sources, including sluicebox vendors and some miners, state that sluices are effective down to 100 mesh (150 μm) and in some cases 200 mesh (75 μm). Certainly, any statements regarding sluicebox recovery must take into account the size and shape of the gold, nature of the deposit, type of sluice used, and method of operation. Seldom

have any quantitative recovery data on gold sluicing been published (1, 2, 3). This absence of performance data is due primarily to the extreme sampling and assaying difficulties inherent in gold sluicing.

The authors have been interested in the concentrating effectiveness and mechanisms of sluicing. The Klondike Placer Miners Association and the Yukon Chamber of Mines provided support and encouragement for a research proposal that was submitted to the Department of Indian Affairs and Northern Development for the purpose of studying the fine gold recovery of sluices. This report is a result of their decision to provide financial support through the Yukon Chamber of Mines.

2. Project Description

2.1 Purpose

The purpose of this research project was to investigate the recovery of fine gold (-20 mesh or -850 μm) in sluiceboxes. It was proposed that this be accomplished by processing a large sample of screened Yukon placer gravel, seeded with known amounts of sized placer gold, in a pilot-scale sluicebox under a variety of closely controlled operating conditions. It was decided to use the same sample of gravel and gold for each test run so that comparisons of recovery data would not be influenced by variation in either the gold or gravel characteristics. This experimental design required a closed circuit facility able to process placer gravel at solid and water mass flow rates typical of modern placer gold mines in the Yukon Territory. Limitation of space and conflict with undergraduate laboratory usage in the Mineral Processing Laboratory at the University of British Columbia resulted in the test sluice facility being constructed at Western Canada Hydraulic Laboratory in Port Coquitlam, B.C.

The majority of placer mines in the Yukon Territory use the Imperial System of measurement for descriptive and comparative discussion of sluicing. The units most commonly used are inches (in)

or feet (ft), time in seconds (sec) or minutes (min), gradient in inches per foot (in/ft), water flow rate in United States gallons per minute (USGPM), solid flow rates in pounds per minute (lbs/min), and suspended solids in parts per million by weight (ppm). The System International (SI) unit equivalents have been included in brackets where appropriate. All descriptions of particle size are expressed in Tyler mesh size with the appropriate equivalents in microns (μm) in brackets.

2.2 Placer Gravel Sample

The gravel used for this pilot sluice test was obtained from the Teck Corporation Granville Joint Venture operation on Sulphur Creek, Yukon Territory. Approximately 15 tons (13.6 tonnes) of gravel were loaded into 34 drums for shipment to Vancouver. The gravel was obtained from a section of the active mining cut where gravel was being mined on July 22, 1985 approximately one foot above bedrock. The gravel was obtained at this elevation above bedrock to obtain a sample of typical gold bearing ore that did not contain fractured bedrock. The assumption that this sample was gold bearing was subsequently borne out during pretreatment of the ore.

Bedrock was avoided to eliminate one variable during testing. The behavior of a test-sluice using well rounded alluvial gravels was selected to be evaluated before considering the complicating effect of various amounts of bedrock material.

The sample was mined with the scrapers used for production mining at Sulphur Creek. The sample was dumped in a segregated pile and a backhoe was used to transfer material from this pile to barrels. Each barrel contained approximately 900 lbs (409 kg) of moist gravel. The sample consisted primarily of well rounded clasts of quartz and chlorite schist less than 4" (10.2 cm) in size. There was also a considerable proportion of fines in the sample. The Sulphur Creek gravel sample, as shipped to Vancouver, was approximately:

40% + 3/4 inch (+19100 μ m)
70% + 1/4 inch (+6350 μ m)
85% + 35 mesh (+417 μ m)
90% + 200 mesh (+74 μ m)

The gravel was dry screened at 1/4" (6.35 mm) on a pilot plant screening facility located at B.C. Research. Approximately 7500 pounds (3400 kg) of -1/4" material was obtained and used for the initial test runs. The +1/4" oversize fraction was subsequently wet screened at 3/4". The resulting separation resulted in a further 8500 lbs (3860 kg) being added to the test sample for the final series of test runs.

2.3 Gold Sample

The placer gold used for this test was provided by Teck Corp. from their gold recovered at Sulphur Creek. Several ounces were hand sieved to obtain 3 troy ounces (93.15 gms) of each of three selected size fractions. The size fractions obtained were Tyler -20 + 28 mesh (-850 μ m + 600 μ m), -35 + 48 mesh (-425 μ m + 300 μ m), -65 + 100 mesh (-212 μ m + 150 μ m).

The hand sieved portions of gold, weighing 93 grams each were subsequently mechanically screened for 10 minutes on a Ro - Tap Testing seive shaker at UBC. The mechanical screening action provided much better sizing of the samples and resulted in the following amounts of the size fractions being available for testing.

Coarse	-20 + 28 mesh,	80.53 gms.
Medium	-35 + 48 mesh,	80.66 gms.
Fine	-65 + 100 mesh,	88.29 gms.
Total Weight		249.48 gms.

Each of the above fractions contained a small amount of impurities but this comprised a very small fraction of the total sample weight (< 500 mg).

The gold was well rounded and showed considerable flattening of the grains, especially in the coarsest fraction. The gold is typical

of that encountered on Sulphur Creek and many other placer mining operations in the Klondike. Each size fraction exhibited a range of shape factors with virtually all particles observed having one dimension less than the other 2 major dimensions. The fine gold consisted of particles whose thickness perpendicular to the major axes indicate a Corey shape factor⁺ between 0.3 and 0.6. The medium sized gold was considerably flatter. The coarse gold showed extensive flattening with many particles having Corey shape factors less than 0.1. This is shown in Figure 1, where gold particles of each size fraction are presented. Each size fraction was photographed at a 45° angle off vertical under the appropriate magnification to make each particle appear approximately the same size on the film negative (1-1/2 mm on a horizontal line through the plane of best focus). The particles of -20 + 28 mesh, -35 + 48 mesh and -65 + 100 mesh were used in 24 tests. The particles of -28 + 35 mesh, and -48 + 65 mesh gold were not used for testing. The particles of -100 + 150 mesh gold were used once. There is no evidence of excess flattening on other physical changes in the samples of gold used during testing when compared with the unused gold. The difference in shape of the gold fractions reflects the extensive sorting and reworking of these gravels during the erosional history of this area. The presence of predominantly flaky coarse gold with much less flaky finer gold indicates a depositional environment where the gold in any one location tended to be equi-hydrodynamic, that is the conditions that favoured the accumulation of thin flakes of coarse gold would also favour the accumulation of finer gold of a higher shape factor. The relative size of individual particles of each size fraction used is presented in Figure 2.

The gold used in these pilot tests was originally recovered in a sluicibox. This did not appear to introduce a bias to the sample. There is however, a possibility that in-situ fine gold with a low shape factor may have been poorly recovered during production sluicing

$$+ \text{ Corey Particle shape factor } = \frac{\text{thickness}}{\sqrt{\text{length} \times \text{width}}}$$

and extraction. Examination of the gold recovered from the gravel sample prior to seeding with the test gold showed characteristics (regarding the shape factor ranges of each size fraction) similar to that of the test gold.

A more detailed description of the nature of the placer gold sample will subsequently be provided in the Masters degree thesis of James Hamilton emanating from this test program.

2.4 Test Sluice Facility

The test sluice was constructed on a pilot plant scale to reproduce the concentrating environment of a commercial sluicebox. Water flow rates up to 400 USGPM ($4.95 \text{ m}^3/\text{min}$) and solid feed rates up to 1200 lbs/min (550 kg/min) were fed to the 12 inch (0.305 m) wide sluice during the testing. The water and solid flows were closely controlled for each test run.

The project was housed in a warehouse with a concrete slab floor. The space provided measured 35' x 100' with a height in the center long axis of 17' (10.7 x 30.5 x 5.2 m).

An overview of the sluice assembly is provided by the photograph of Figure 3 and schematic drawing of Figure 4.

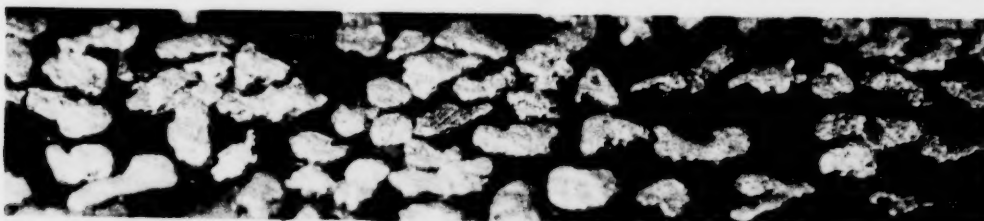
2.4.1 Sluice and Entry Flume

The sluice was 8 feet long and 12 inches wide. The base of the sluice was $3/4$ " plywood and the framework consisted of 2" x 4" lumber. The sluice was lined along its base with abrasion resistant rubber (1/8" - Armourbond) and the sides were sheet acrylic, as shown in Figure 5.

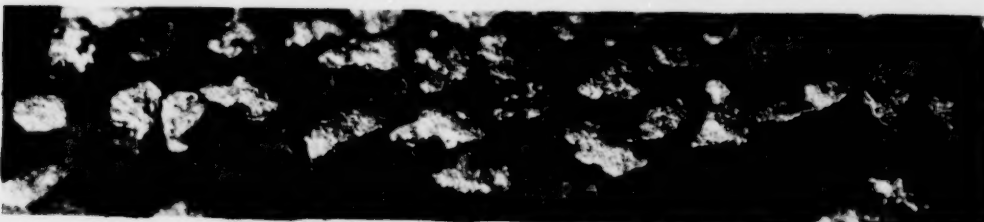
The entry flume consisted of an upwelling-type discharge box for the process water and a 5' long flume. All were constructed of plywood as shown in Figure 6a. The discharge box received water from



-20+28 Mesh (2.0:1) used for 24 tests



-28+35 Mesh (2.6:1) not used



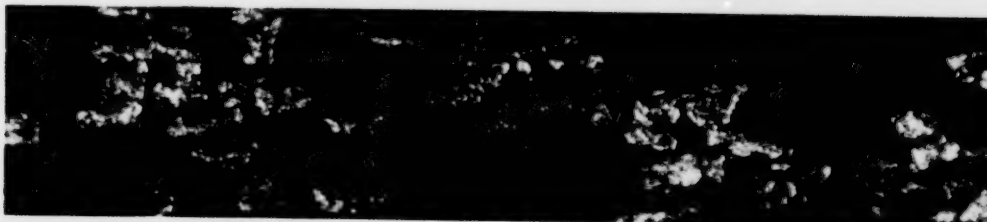
-35+48 Mesh (3.7:1) used for 24 tests



-48+65 Mesh (5.2:1) not used

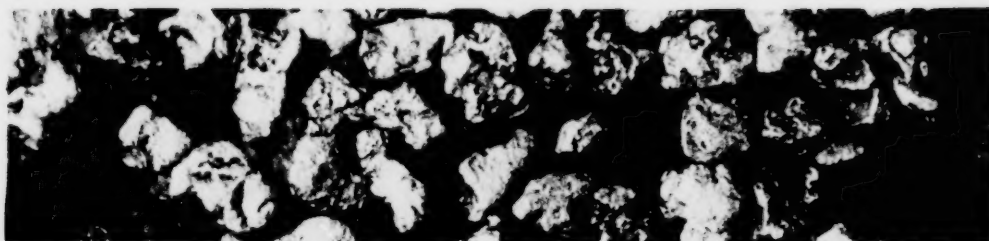


-65+100 Mesh (7.2:1) used for 24 tests

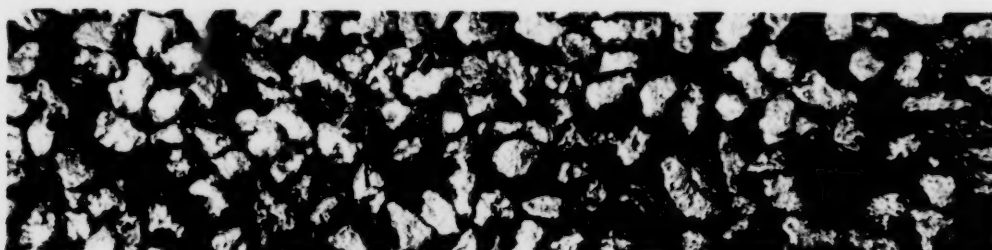


-100+150 Mesh (10.4:1) used for 1 test

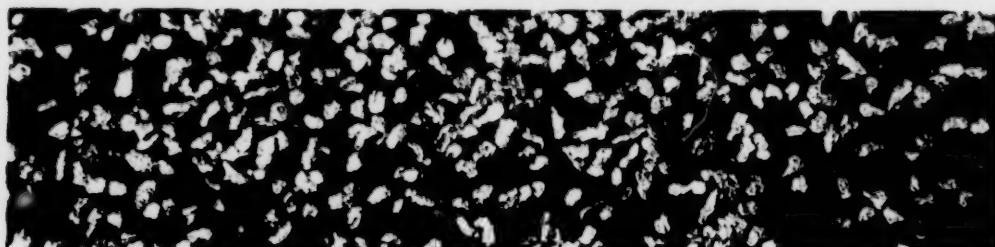
Figure 1 Gold of Various Size Fractions
-photographed on a 2 mm grid
-magnification in brackets



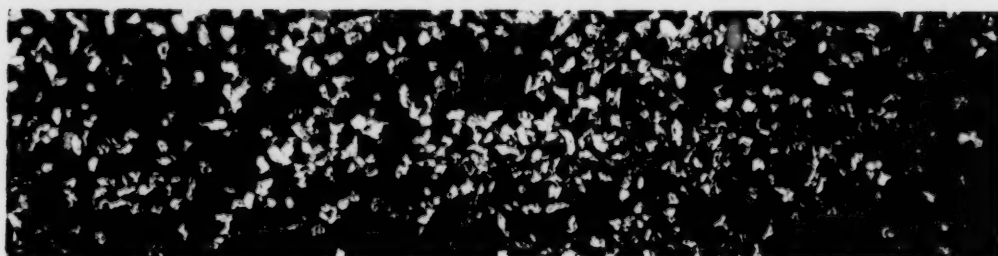
-20+28 Mesh



-35+48 Mesh



-65+100 Mesh



-100+150 Mesh

Figure 2a Gold Particles - Same Magnification

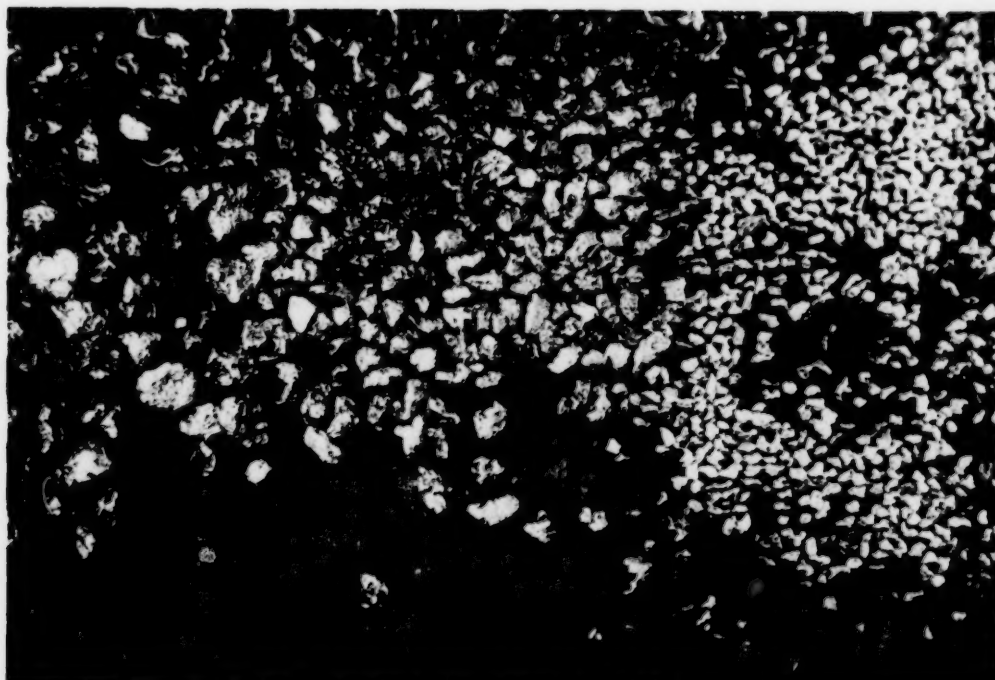


Figure 2b1 -20+28#, -35+48#, -65+100#
(from left to right)

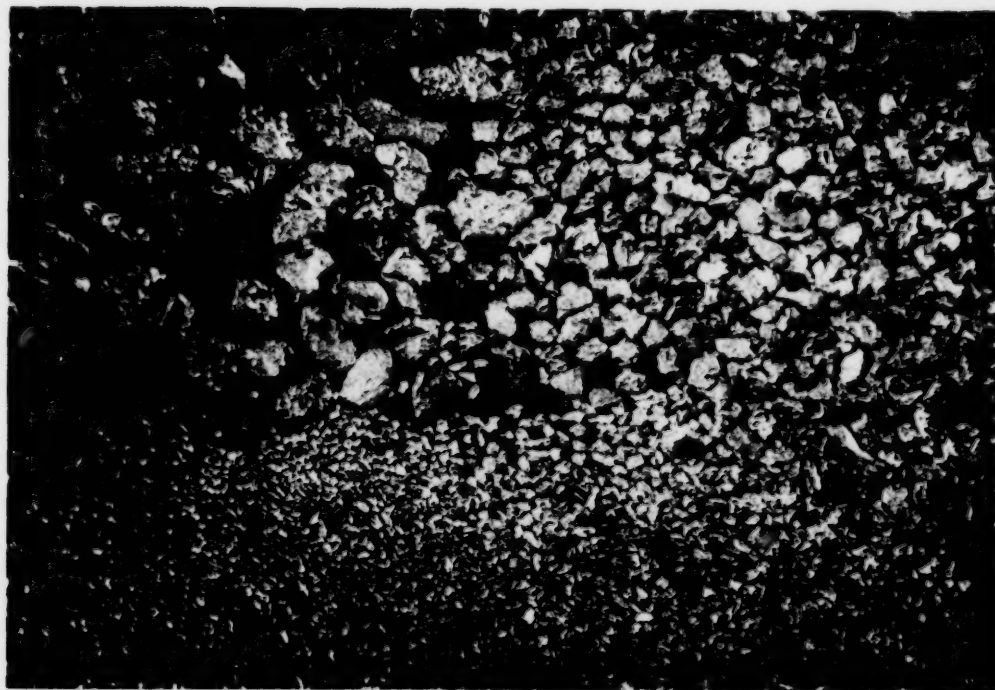


Figure 2b2 -20+28#, -35+48#, -65+100#, -100+150#
(clockwise from upper left)



Figure 5a Sluicebox
- end view

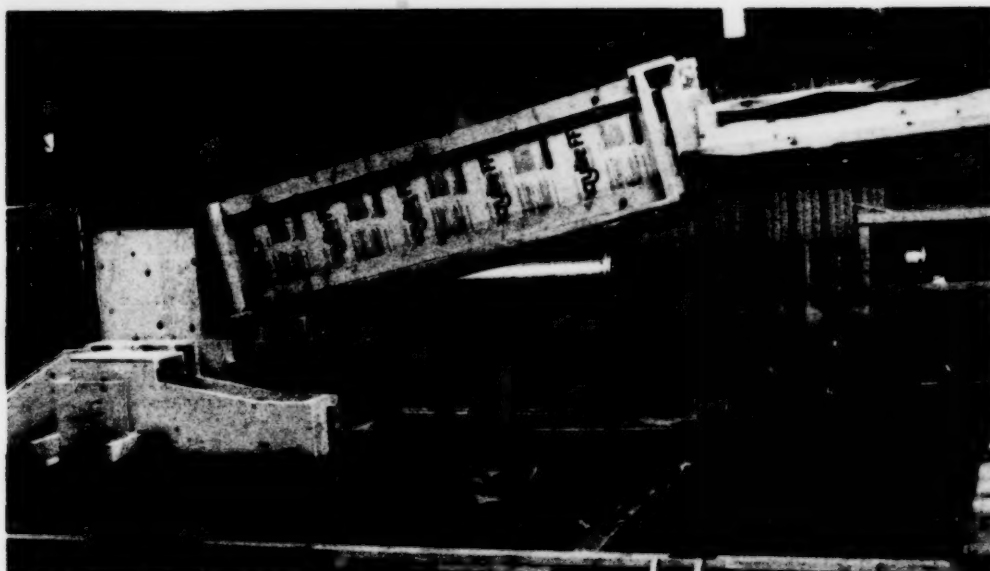
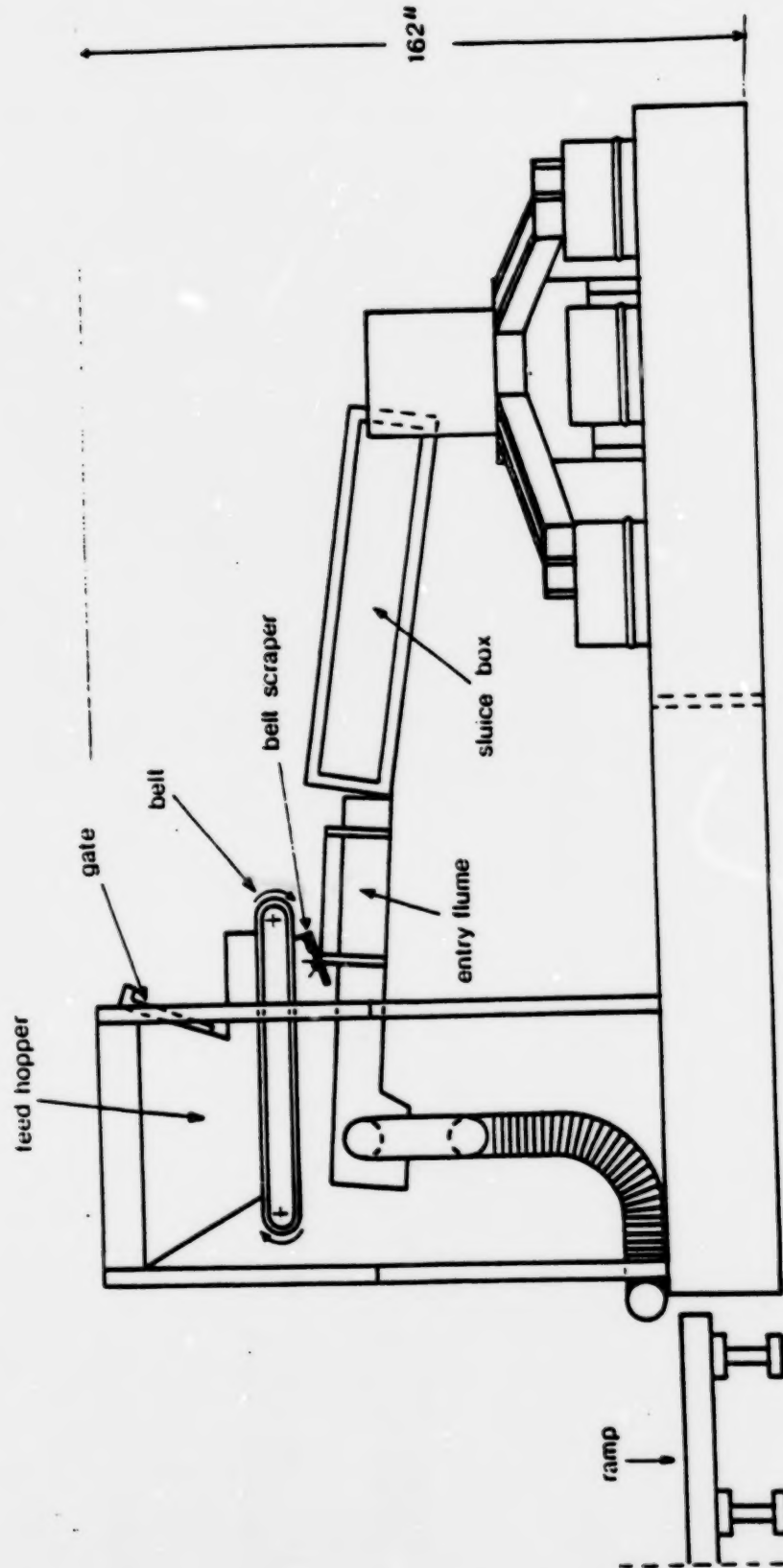
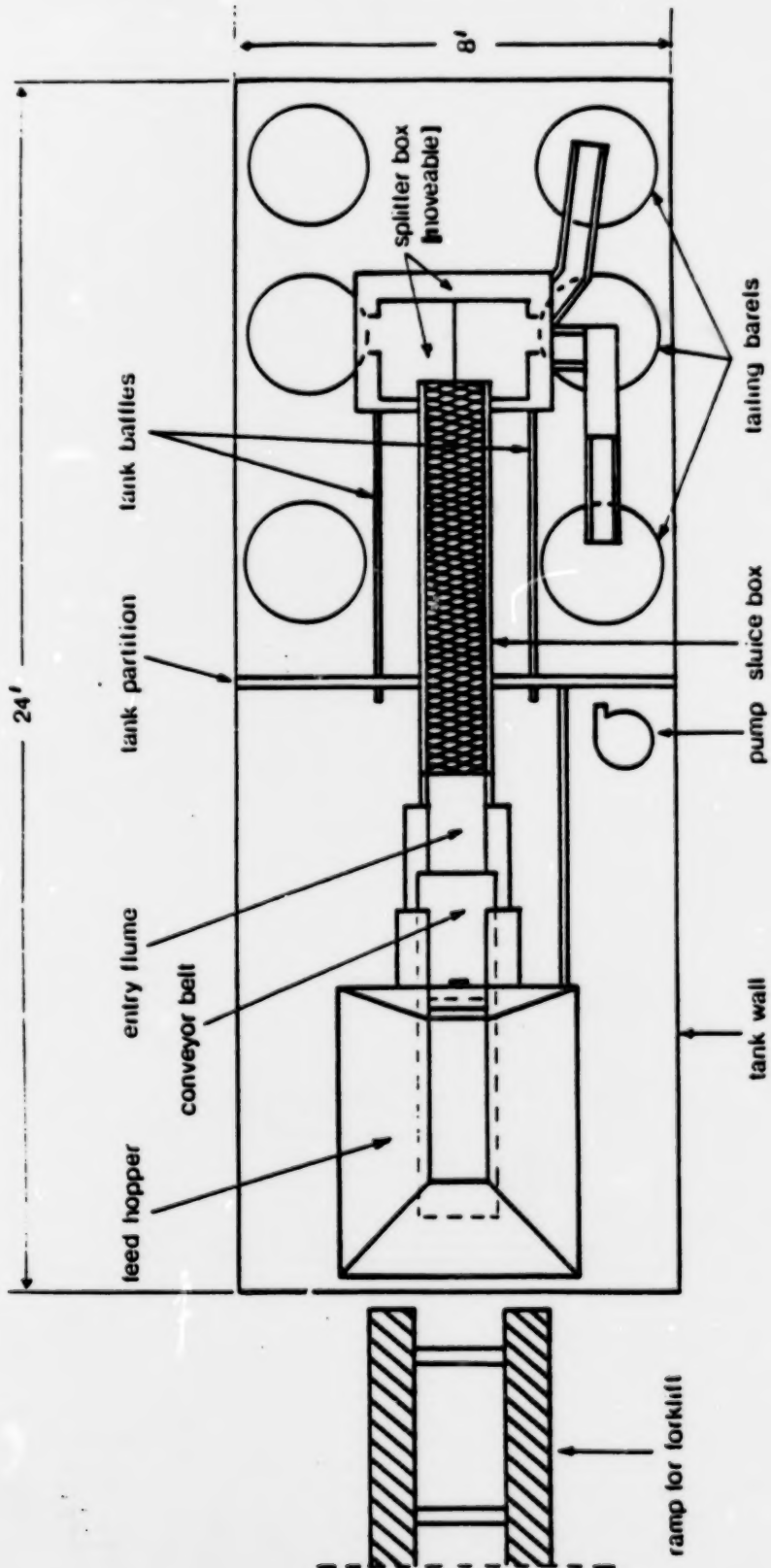


Figure 5b Sluicebox
- side view



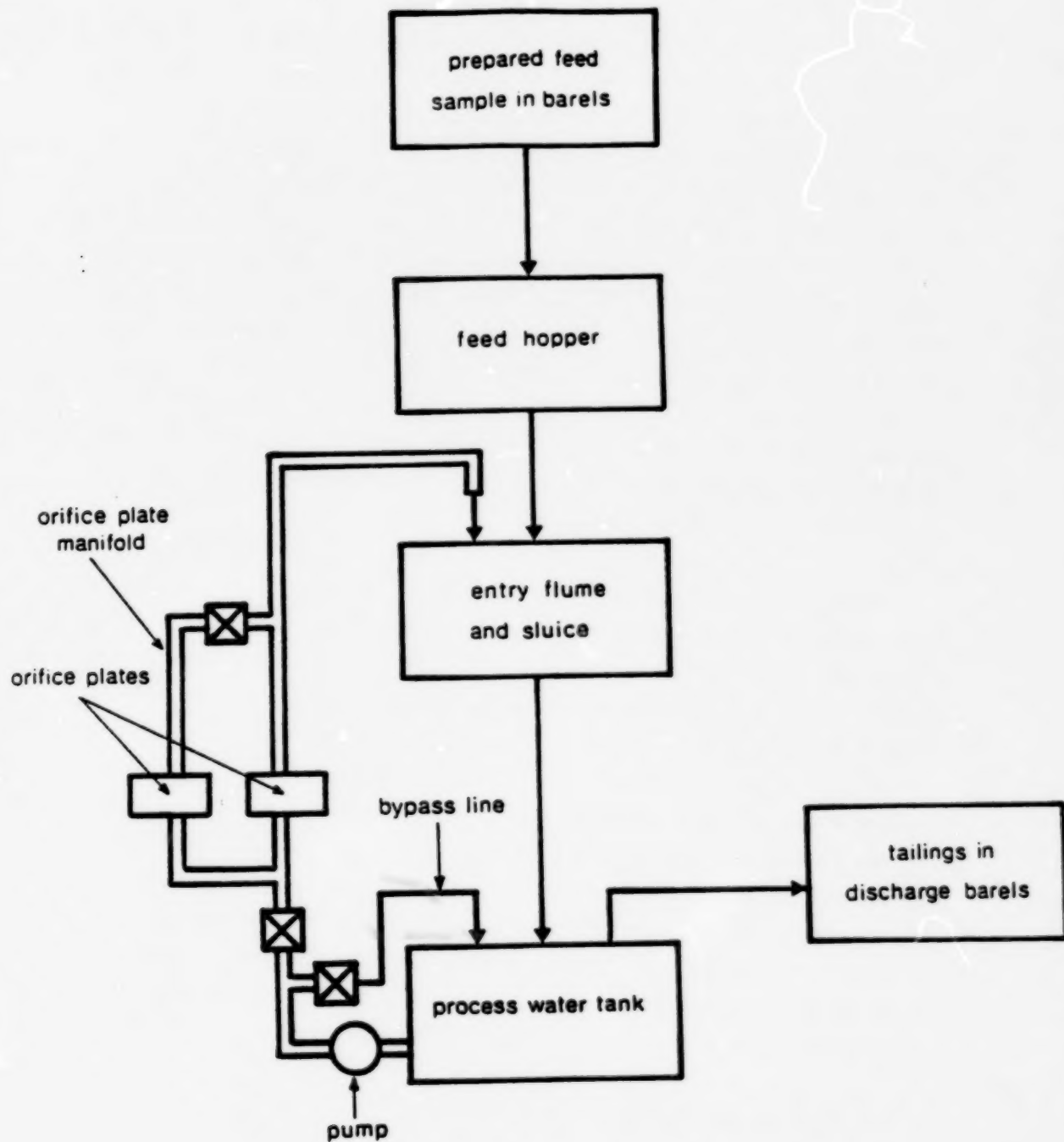
SIMPLIFIED SIDE ELEVATION VIEW

FIGURE 4a



SIMPLIFIED PLAN DIAGRAM NOT SHOWING PIPING DETAILS

FIGURE 4b



FLOW CHART OF TEST SLUICE.

the pump and allowed the velocity of discharge from the 6" (15.2 cm) feed line to dissipate prior to introduction to the entry flume. This arrangement was employed to minimize the effects of water at relatively high velocities entering the flume and to allow a relatively tranquil flow to accelerate down the flume to the point of gravel introduction. The 5' (1.53 m) long entry flume was lined on the sides and base with abrasion resistant rubber. The gold bearing gravel was introduced to the flume approximately 30" (76 cm) from the entrance to the sluicebox. By feeding the test gravel into the entry flume at this point the process water was then flowing with sufficient velocity to carry the coarser aggregate into the sluice and still allow the heavier mineral particles to concentrate along the base of the flume by gravity settling in a dispersed shearing particulate bed that formed at the base of slurry.

The lining of the flume was extended 2" (5.1 cm) into the sluice and fastened by screws and silicone sealant. (Figure 6b) The base of the flume was raised approximately 3/4" (19 mm) above the base of the sluice where they joined, and close to the point where the slurry passes over the first riffles. The adjustable slope of the entry flume was set to minimize the flow transition disturbance at the sluice entry. Visual observation of the slurry behavior at the entry to the sluice was monitored to achieve this. The transition from the relatively smooth surface base of the entry flume to the riffled area of the sluice always caused some flow adjustment by the slurry. Visually minimizing the distance down the sluice that the slurry attained equilibrium of depth and scour behind the riffles was considered adequate for the purposes of this program.

2.4.2 Riffles and Matting

In this report, the word "riffle" is used to describe any device used in the base of a sluicebox to affect concentration of heavy minerals, particularly gold. Many Yukon placer miners consider the term "riffle" to mean "dredge riffles" made up of 1" to 1 1/2" (25 - 38 cm) angle iron.

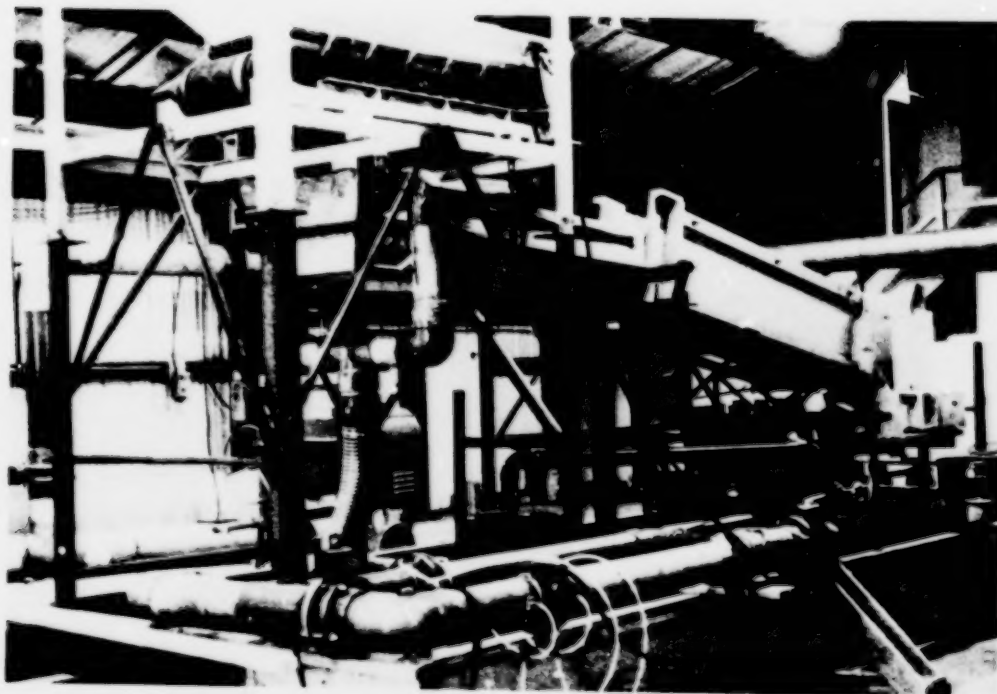


Figure 6a Entry Flume

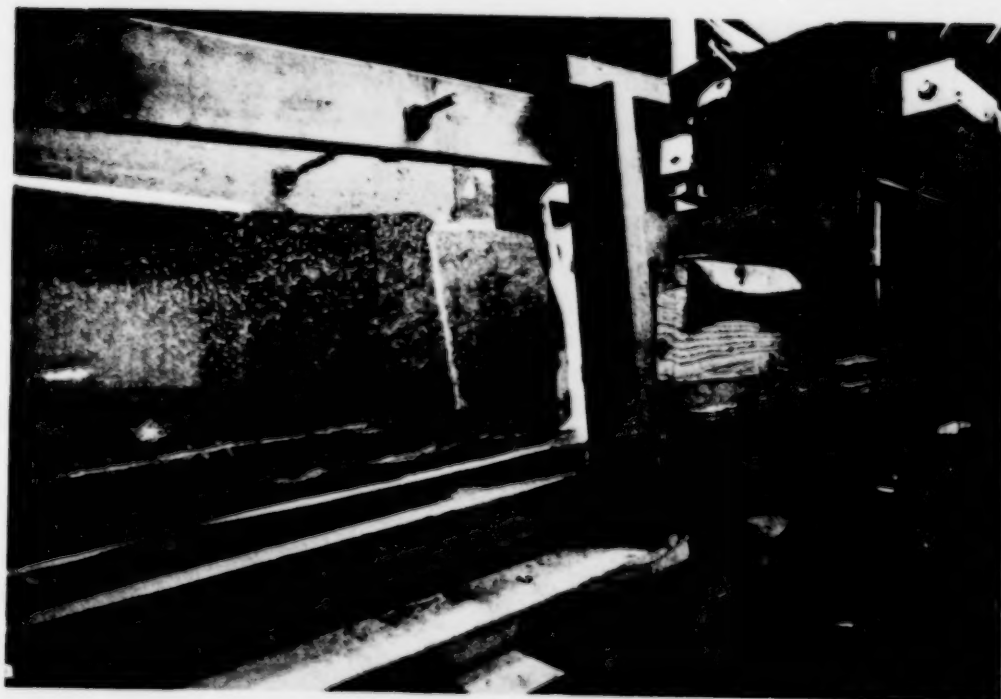


Figure 6b Entry Flume - connection to sluice

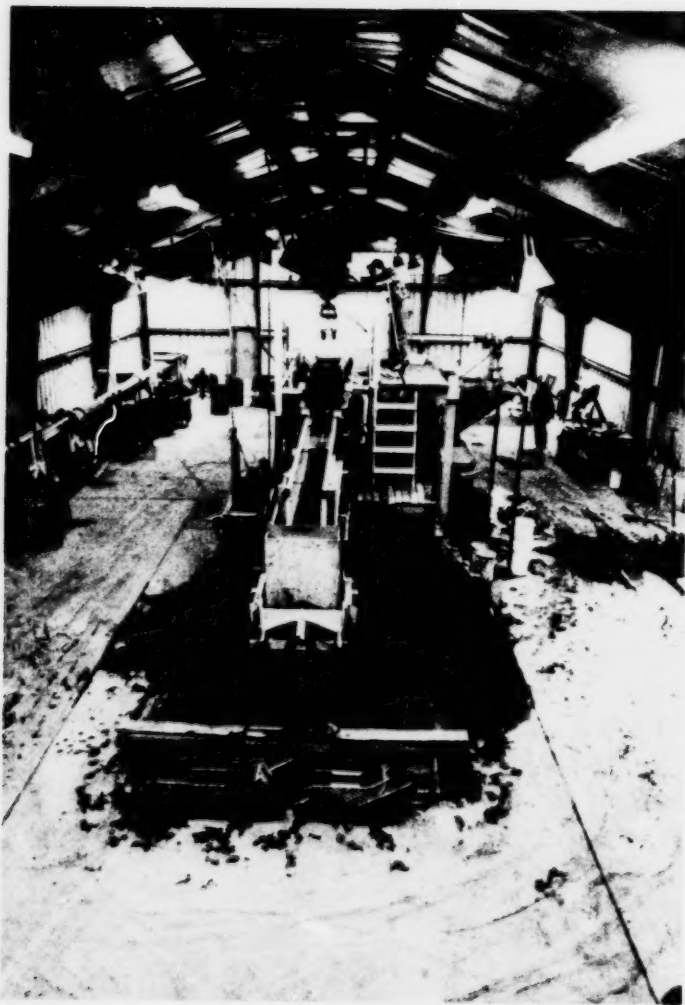


Figure 3 Overview of Test Sluice Facility

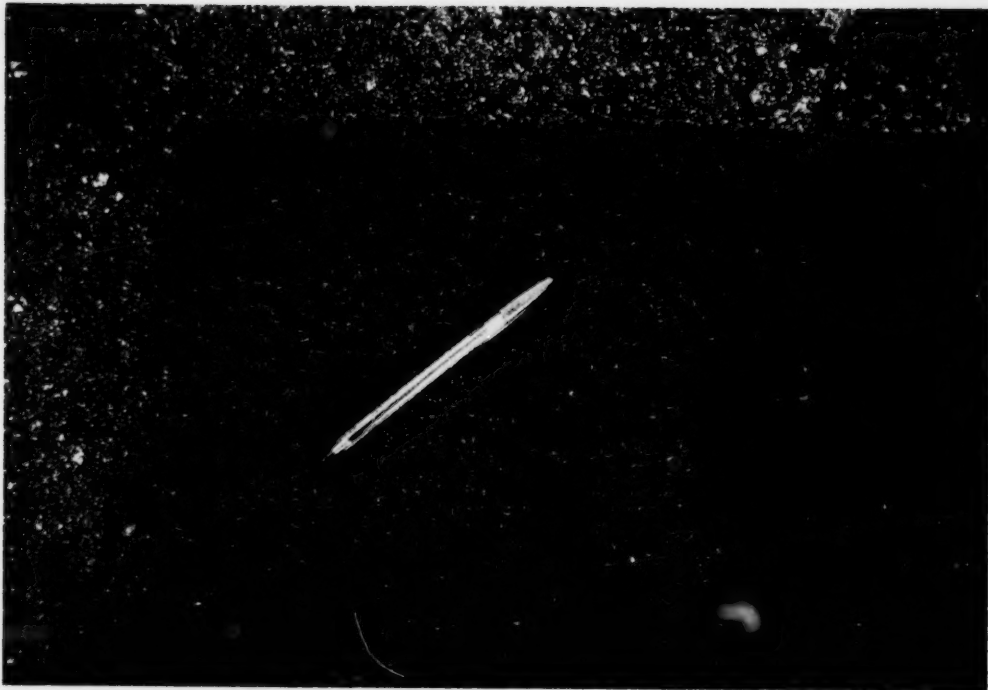


Figure 8a Nomad Matting



Figure 8b Cocoa Matting

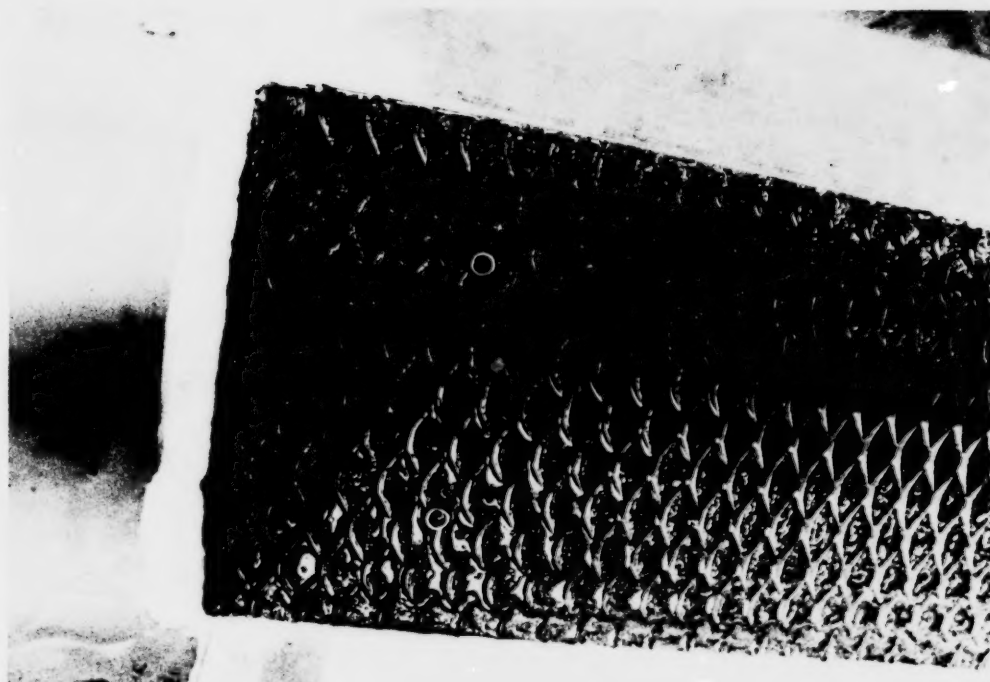


Figure 7a Expanded Metal



Figure 7b Dredge Riffles

The riffles selected for testing were expanded metal (1-10H) and dredge riffles (1 1/4" angle iron). Figure 7 shows photos of these two types of riffles. These are the two types of riffles employed almost universally in the Yukon. There are many different types of expanded metal and angle iron is used in sizes from 1" to 4" (25 - 100 mm) . The two configurations selected for testing are the most common for treating slurries that have had coarser aggregate, + 1/4" to 1", (6.4 - 25 mm) removed by screening, trommeling, passing over punch plate, or use of a Derocker.

The matting selected was Nomad matting (Figure 8a) which has recently found widespread use in the Yukon. Figure 4 shows the nature of this matting. The unbacked, 3/8" (9.5 mm) thick, blue version of this product in addition to being quite popular in the Yukon also fulfilled the main requirements of matting in a sluicibox. These are that the matting be: easy to clean, able to prevent flow under the riffles, durable, and provide a sheltered environment for heavy minerals to collect should scour expose the matting.

Cocoa matting (Figure 8b) was used for one test to determine and compare its ability to collect gold when used in a high scour environment with expanded metal. It was not used again due to the extreme difficulty in completely removing the gold from the matting during clean-up. Quick and efficient gold removal was a necessity for this test program.

The matting was cut into sections 12" wide x 24" long (305 x 610 mm) to facilitate cleaning the sluice out at intervals 0-2, 2-4, 4-6, 6-8 feet (0-.305, .305-.610, .610-.915, .915-1.22 m). The expanded metal was also cut into 4 sections 12" x 24". The dredge riffles were fabricated in 3 sections 24" long, with a 4th section 8" (20 cm) long. Only the data from the first 3-24" sections were used for the dredge riffles. The 8" section was always placed furthest from the head of the sluice. The purpose of this section was to prevent the unusual scour conditions (that occurred as the slurry accelerated for the drop to the sluice base at the last riffles) from occurring at the end of the 4-6' test section.

The sections of riffle and matting were always placed in the same location of the sluice. That is, the expanded metal or dredge riffle section, with it's associated matting, used at head of the sluice, or #1 position, was always used in this position.

2.4.3 Pumps and Tankage

The water supply system used is shown in the photos of Fig. 9. A 4" (10.2 cm) submersible Grindex pump was used for the process water pump. The pump discharge was connected to a control manifold with two outlets, both regulated with valves. A 3" (7.6 cm) gate valve and discharge line was used to bypass pump output to a return line into the circulating tank. A 4" discharge line, controlled by a butterfly valve, was placed immediately before the orifice plate manifold.

Water flow rate was monitored by use of manometers connected to orifice plates in the orifice plate manifold. The manifold was constructed so that the flow could be isolated through one orifice plate or could flow through both orifice plates at once. The orifice plate manifold control valve was set in the closed position to route the entire flow through one orifice plate. This setting was employed when low flow rates were used as the entire flow was only sufficient to give significant pressure drop across one orifice. At high flow rates the control valve was set to open. This allowed the flow to be measured across two orifices in parallel and was used when the pressure drop across the single orifice exceeded 25" (63 cm) of measuring fluid. The U-tube manometers were capable of measuring approximately 35" (89 cm) of fluid head but any fluctuations at this head could cause measuring fluid loss. Red Meriam fluid, specific gravity 2.95, was used in the manometers.

The discharge from the orifice plate manifold flowed directly to the discharge box of the entry flume as shown in the photo of Figure 9b. Six inch flexible hose was used here to accommodate the various slope adjustments of the entry flume.

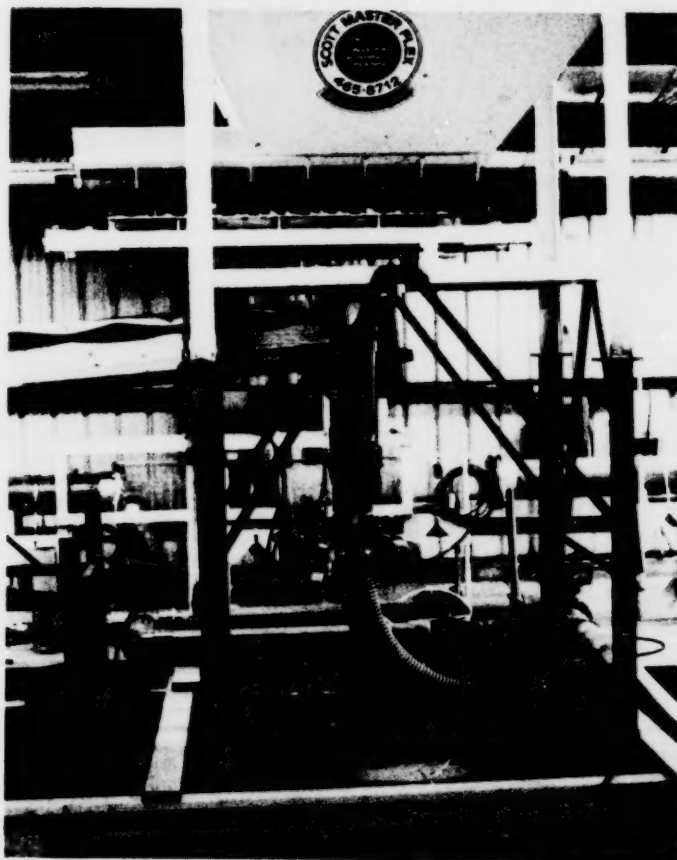
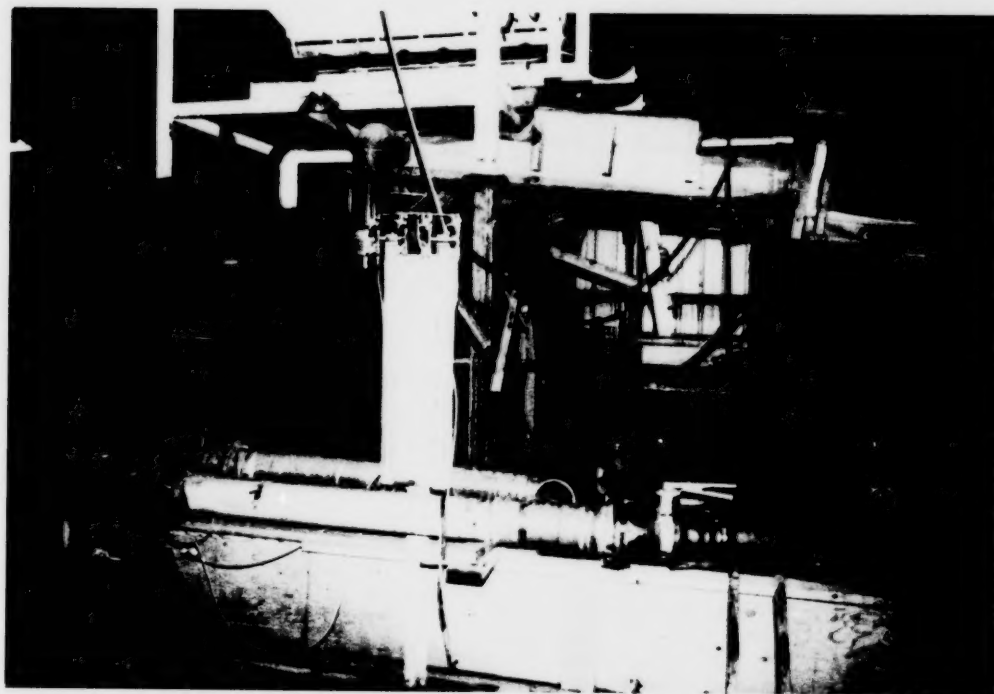


Figure 9a Piping and
Manometers
-showing, from r to l,
3" bypass line and
valve, 4" butterfly
valve, orifice plate
manifold, U-tube
manometers with
connections to
pressure ports

Figure 9b Piping
-view during
construction of
tank and piping

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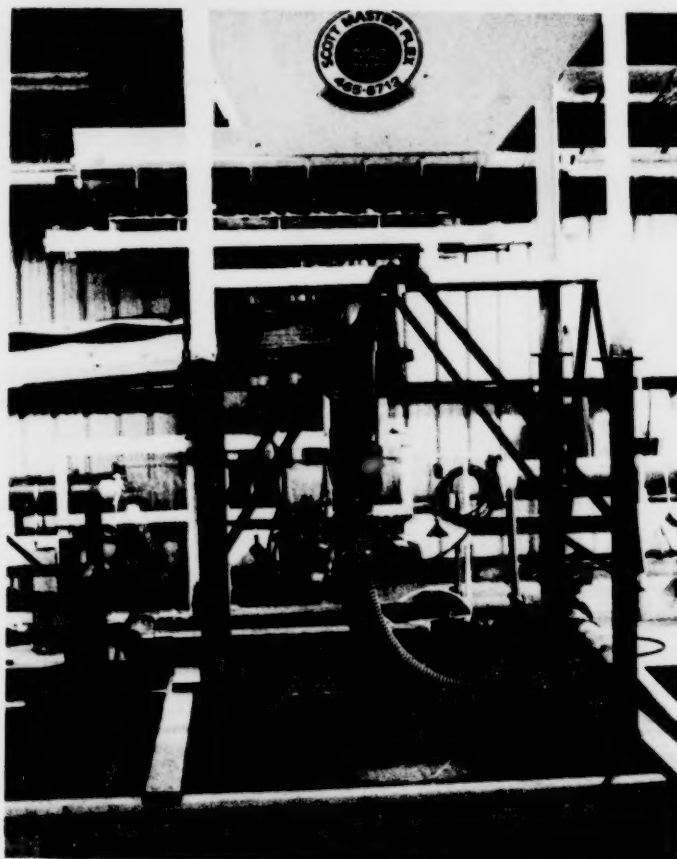
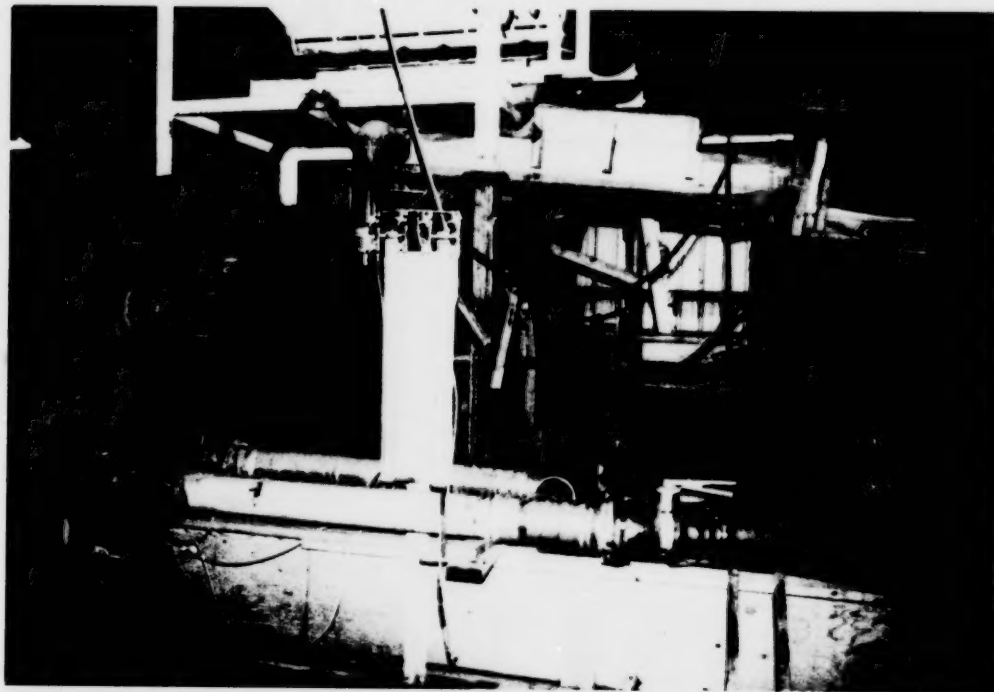


Figure 9a Piping and
Manometers
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3" bypass line and
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Figure 9b Piping
-view during
construction of
tank and piping

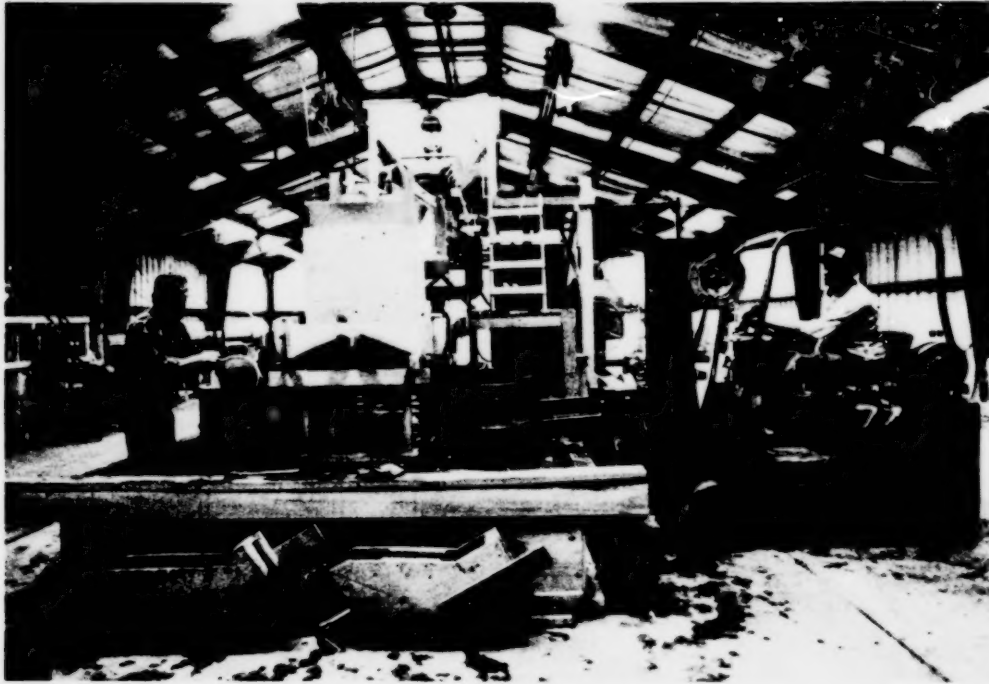


Figure 14a Removing Tails

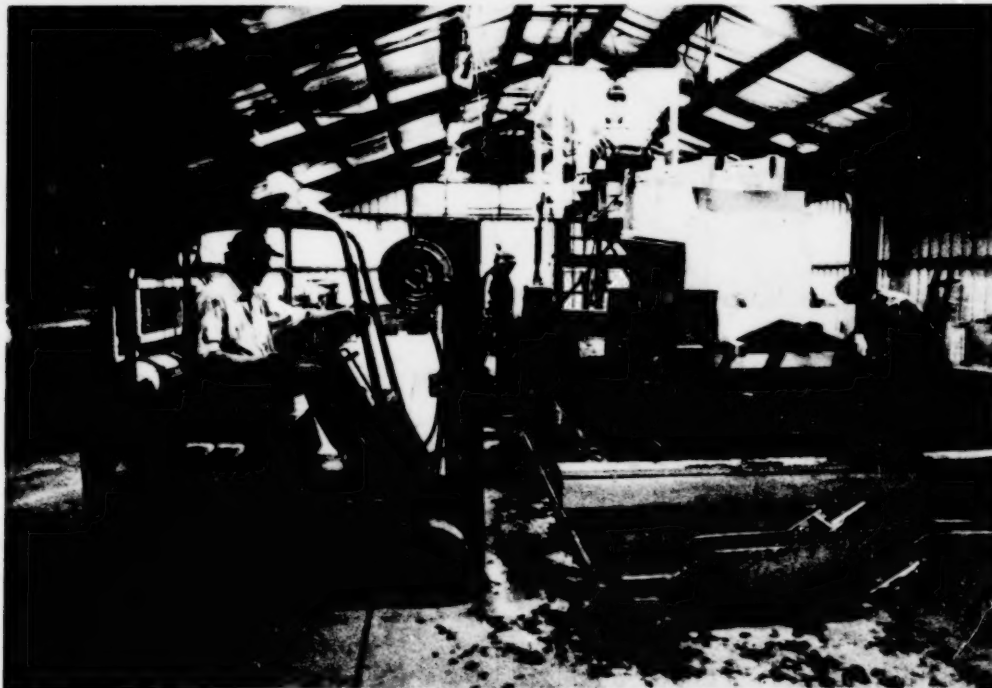


Figure 14b Removing Tails

The process water was completely recirculated. The volume of water used for each test run was approximately 2000 US gallons (25 m^3). This was contained in the process water tank which measured 24' long, 8' wide, and 1 1/2' deep ($7.3 \times 2.45 \times 0.46 \text{ m}$). The entire sluice system including the gravel feed hopper, was supported on legs that stood inside the tank so that most spillage would be contained within the tank volume. The tank was partitioned at the center of its long dimension. The portion of the tank where the discharge from the sluice was distributed into barrels was termed the "discharge compartment". Ten foot (3.05 m) long baffles were placed in this tank perpendicular to the tank partition so that they formed a 10' long 2' wide (.61 m) channel in the center of the discharge compartment. The barrels for receiving the sluice discharge were placed between these baffles and the tank walls. This arrangement forced the overflow from the discharge barrels to flow to the end of the tank, opposite the partition and then along the 2' wide channel to an overflow weir cut in the tank partition where it discharged into the other half of the process water tank. This was called the "suction compartment" because it contained the main process water pump. An 8' baffle was placed in this tank 2' from one side and perpendicular to the partition. The pump was located at the closed end formed by the baffle, tank wall, and partition. This was to prevent short circuiting between the discharge compartment overflow and the main pump. The features described can be observed in several figures (Figures 3, 4, 9).

Upon completion of each test run, the discharge compartment was drained by a pump in order that all the settleable solids that had overflowed from the barrels could be recovered. A 1 1/2" (3.8 cm) submersible pump was located in the center channel of the discharge compartment near the overflow weir to pump the process water in the discharge compartment to a "holding tank". This tank was a 4' x 5' x 3' ($1.2 \times 1.5 \times 0.9 \text{ m}$) deep steel tank to which 4' (1.2 m) plywood sides were added, giving a total depth of 7' (2.1 m). Water in the holding tank could be returned to the process water tank via a drain pipe with a globe valve which discharged into the suction compartment.

2.4.4 Materials Handling

The experimental design of this project required the development of a unique materials handling strategy. Considerations that governed the evolution of this strategy were:

1. The sample must be handled in individual quantities that are:
 - a) as large as possible to minimize the number of containers
 - b) small enough to be handled at relatively low cost
2. The sample must be completely recovered for reuse during the next test.
3. The sample must be remixed to attain a homogeneous distribution of all particle types and sizes, including gold, prior to reuse.
4. The sample should be fed to the sluice with a uniform mass flow rate and minimal process interruptions.

These considerations, together with the cost of the methods to achieve them led to the selection of the following equipment for materials handling.

A. Barrels

A fundamental aspect of the experimental design was the size of the containers for the gravel sample. Ideally as few containers as possible would have reduced the amount of time and effort to handle them. Large containers imposed high handling costs due to the need for expensive, specialized, large machinery. It was decided to use standard 45 gal barrels as the containers. They were inexpensive (\$5 Can each) and each could accommodate approximately 900 lbs. (410 kg) of saturated gravel.

The placer sample was shipped from the Yukon to Vancouver in thirty four (34) heavy duty open barrels with lids. Most of these were used as containers during test runs. Twenty five (25) additional barrels were purchased from a local steel drum recycling plant. These were heavy gauge barrels that had sustained damage to their tops. The tops were cut off and the edges rolled over. These drums were made of lighter gauge steel than the barrels used for shipping, and did not have large ribs rolled in them. As a consequence they were more difficult to handle and easier to damage. Though all barrels showed signs of hard use only 6 were damaged beyond further use during the tests.

B. Feed Hopper

A feed hopper and well sealed, variable speed conveyor belt feeder were selected to provide closely controlled solids feed to the pilot sluice. A 2 cu. yd. (1.67 m^3) hopper with a variable speed 12" wide belt base and fully adjustable gate opening was made available by Western Canada Hydraulic Laboratory in Coquitlam, B.C. This hopper, plus a facility to house the project were rented from WCHL for a nominal fee.

The hopper was rectangular in plan and supported by 4" x 4" (10 x 10 cm) angle iron legs at each corner. Extensions were added to the legs so the top of the hopper stood 162" (411 cm) off the floor (see Figures 5a, 9b). This provided enough clearance under the conveyor to accommodate the vertical requirements of the sluice and discharge system and yet leave enough clearance above the hopper for dumping the sample barrels. The conveyor belt was supported on idlers throughout its length and driven by a hydraulic motor. Speed control was achieved by using a control valve to regulate the amount of hydraulic fluid supplied to the motor. An electrically driven pump provided hydraulic pressure. Figure 9b shows a side view of the hopper belt feeder assembly.

C. Forklift

A Baker FJD-040 propane powered forklift with a modified paper roll clamp was used to handle the barrels of gravel (Figure 10a). The forklift was fitted with a high lift mast, capable of lifting a barrel up to a 145" (368 cm) clearance above its base.

The clamp was originally used to handle 4000 lb (1820 kg) rolls of paper in a printing shop. The clamp was modified by cutting off the roll clamp arms and fitting specially fabricated clamp arms for the barrels. These arms were designed to grip the barrels in the center between the ribs. The ribs prevented the barrels from dropping out of the clamps when closed lightly against the barrel.

The clamp was fully rotational in a plane perpendicular to the center line of the forklift. The barrels were emptied by rotating the clamp 180°, thereby turning the barrels upside down. The full height extension of the mast was not sufficient to enable a barrel to be lifted from the floor level to a height that would allow the base of the barrel to clear the top of the feed hopper. A ramp was constructed of steel and wooden planking to provide the additional 20" (51 cm) of elevation required. Figure 10b shows the fork lift dumping a loaded barrel into the feed hopper.

D. Mixer

Prior to each test run, the placer gravel sample and gold had to be remixed. During test runs there was considerable overflow of fines into the discharge compartment. Though these solids were recovered, they had to be remixed with the sample. The placer gold recovered had to be evenly redistributed in the sample as well.

An Atika cement mixer (Figure 11) was purchased for remixing the sample. It had a practical capacity of approximately 2/3 barrel. A hopper was constructed to receive the discharge from a sample barrel

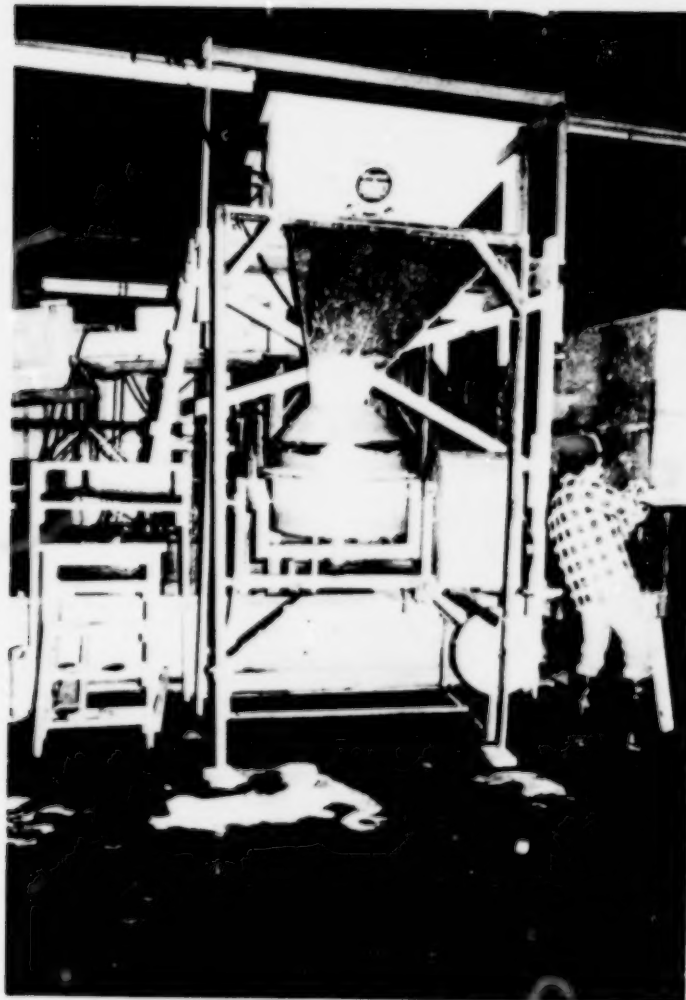


Figure 11 Cement Mixer

-note rubber skirt extending
into mixer to prevent spillage

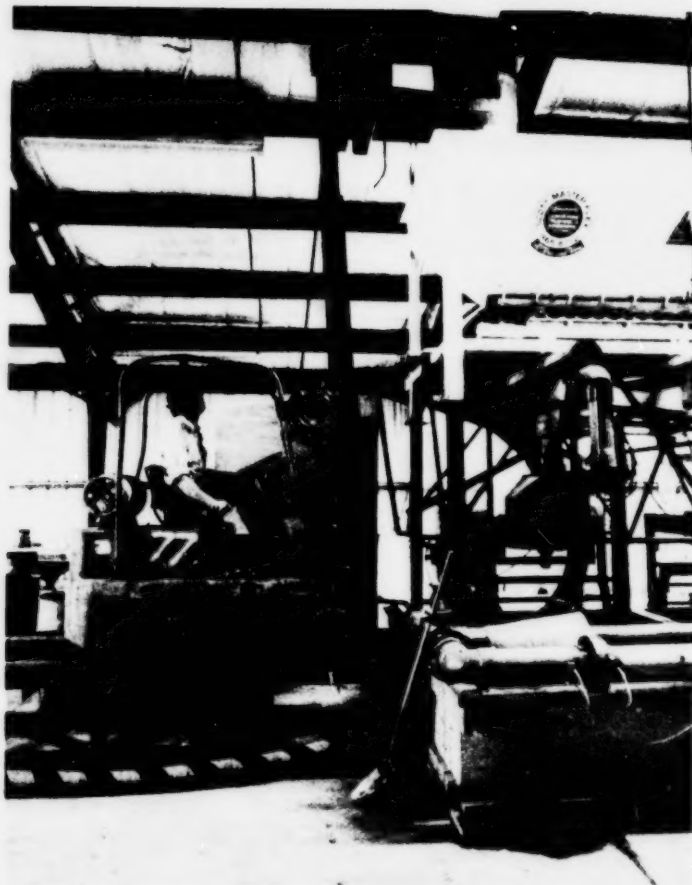


Figure 10a Forklift
- loading hopper



Figure 10b Forklift

and feed the gravel into the mixer. The forklift was used to feed gravel to the mixer and then remove barrels of reconstituted feed gravel.

E. Discharge System

Six sample barrels could be accommodated at one time in the discharge compartment of the process water tank, arranged in two groups of three (see Figures 3, 4a, 12). Each group was placed, evenly spaced, in two areas between the baffles and tank walls. A rubber lined plywood box, the "splitter box", was placed at the end of the sluice to split the discharge into two equal portions. A steel plate aligned with the center line of the sluice was installed in the splitter box to accomplish this. The two slurry streams exited symmetrically from the sides of the splitter box at right angles to the axis of the sluice. A system of plywood troughs directed each slurry stream to one of the three barrels on each side. The sluice could be operated continuously during the filling of all six barrels in the discharge compartment.

F. Secondary Extraction & Weighing

The concentrates recovered from each 2' test section of the sluicebox were first hand sieved using full height 8" (20.3 cm) brass sieve screens. Subsequent extraction of pure native gold was carried out on a Goldhound spiral bowl concentrator (Figure 13). The concentrated placer gold of the selected size fractions had as many impurities removed as possible by selective panning and by using hand magnets. The samples were dried in an oven at 275 °F and then weighed on a Mettler 440 digital top loading balance.

3. Procedure

3.1 Preparation of Gravel Sample

The gravel sample was initially processed by dry screening at 1/4". This was to prevent losses of clay and silt fractions that would have occurred while wet screening 15 tons (13.6 tonnes) of gravel on a 1/4" (6.35 mm) screen.

The 1/4" oversize was later wet screened at 3/4" (19 mm). During this screening, a 1" (25 mm) water hose with a nozzle, controlled by a ball valve at the nozzle, was used to wash the gravel as it passed over the screen. The water was used in sufficient quantities to ensure all the oversize was washed clean of adhering fines. Care was taken to ensure that all settleable solids were retained with the screen undersize.

Upon conclusion of each screening operation the underflow material was shipped to the test sluice facility in Port Coquitlam, British Columbia.

3.2 Stripping Runs

A total of 29 complete test runs were conducted during testing. Four of these runs (#1, 2, 3, and 15) were used to remove native gold from the gravel prior to seeding in three sizes of gold. The -1/4" gravel was run through the test sluice three times at the start of the program. This was to ensure the naturally occurring gold was completely removed and to allow appropriate sluice operating procedures to be developed. Problems encountered during the first two runs were solved by slight modifications to the apparatus or procedures. The almost negligible amount of gold recovered during the third test indicated that very little natural gold remained in the -1/4" sample.

Test run #15 was conducted on the underflow from the wet 3/4" screening. There were considerable fines that had adhered to coarser aggregate in this material during the dry screening at 1/4". The

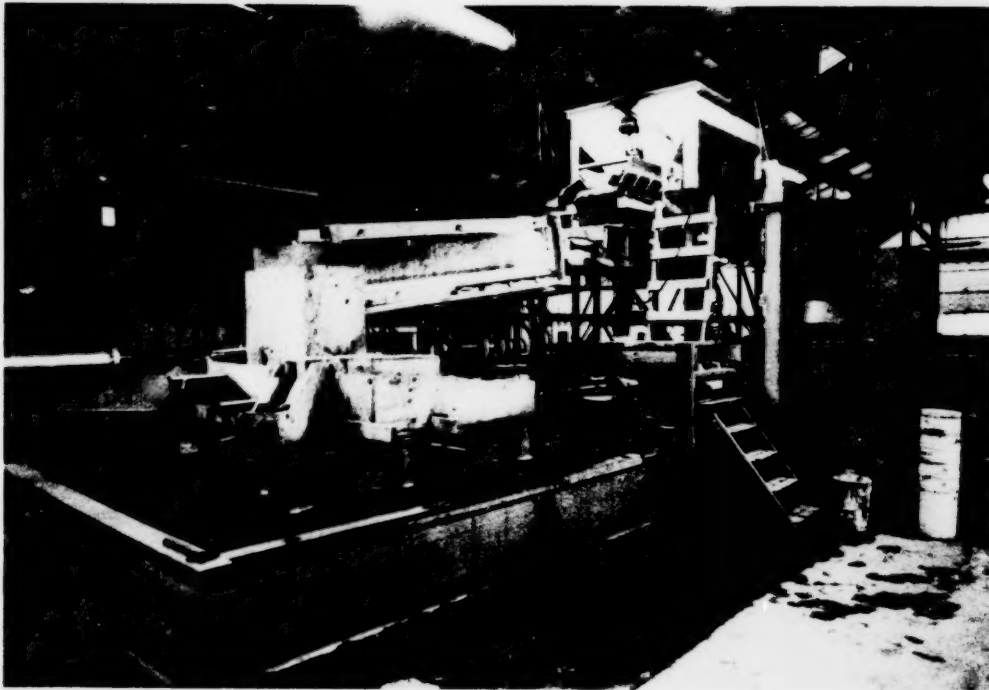


Figure 12a Discharge System

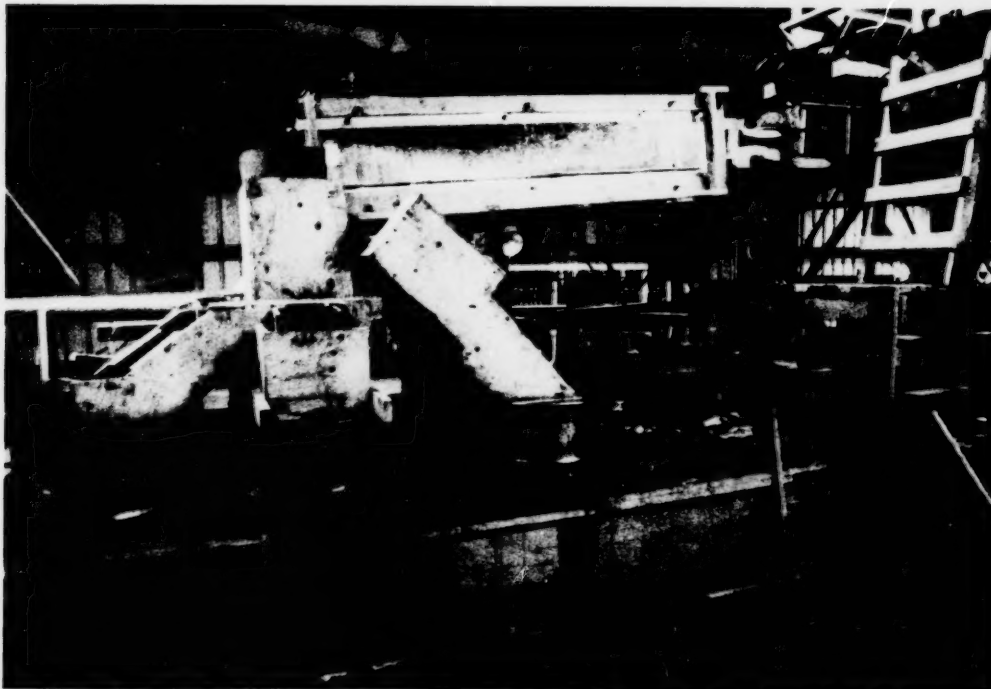


Figure 12b Discharge System

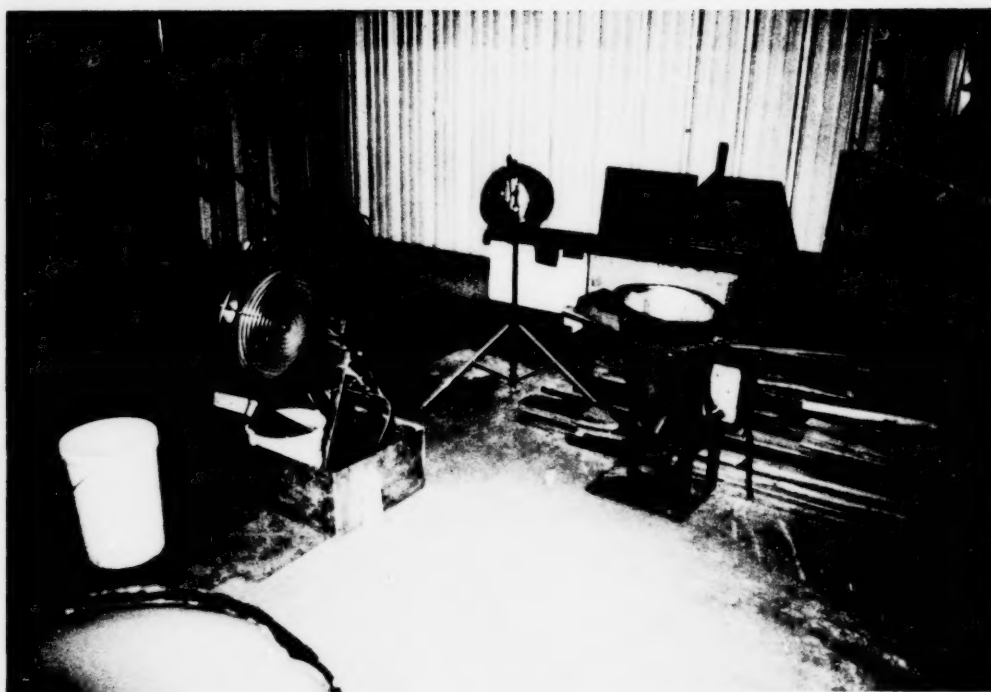


Figure 13 Goldhound and Syntron Feeder

presence of fines indicated similarly sized gold could also be present. This material was sluiced to remove the native gold. The amount of gold recovered in the sluice after this run showed that a significant portion of fine gold had adhered to the oversize from dry screening. Based on prior test results, it was concluded the single #15 stripping run was sufficient to clean out the natural gold.

The procedures used during the stripping runs were almost identical to those of the subsequent test runs. The only difference was that the gold extracted during the stripping comprised all sizes and required slightly different techniques for extraction. There was no mixing of gold with gravel prior to the stripping runs. At the conclusion of runs #2, 3, and 15 the fines that overflowed the barrels in the discharge compartment were recovered for remixing prior to the next test.

3.3. Reconstitution of Test Gravel

Upon completion of each test run, settleable particulate solids were found in the following locations:

1. Sluicibox riffles - The material retained here, typically termed "concentrate", was subjected to further treatment to extract the placer gold for sizing and weighing.
2. Tailings Barrels - The coarse aggregate and some of the fines were reclaimed in the barrels placed in the discharge compartment. The majority of the solids were recovered in the barrels.
3. Discharge Compartment - Some solids always overflowed from the barrels and settled in the discharge compartment. If the barrels were allowed to overfill (more than 2/3 full of solids) considerable overflow of solids could occur. The amount of material that overflowed during each test

depended primarily on the slurry flow rate through the sluice. At high flow rates the fluid velocities in the barrels were sufficient to carry over fairly coarse particulates, especially when the barrels were more than 1/2 full of solids. For tests using low flow rates the overflow solids consisted of a small amount of silt and fine sand.

4. Suction Compartment - Material reaching this compartment was very fine grained, consisting primarily of clays and fine micas. This material was treated as part of the process water.
5. Accidental Spillage - There were, inevitably, some accidental spills of solids onto the floor surrounding the water tank. Every effort was made to minimize these occurrences. Any spillages were recovered by a wet vacuum cleaner and remixed with the test gravel.

The concentrate was removed from the sluice immediately after each test run. The discharge system was washed down and removed. The tailings barrels that received the final portion of the test gravel were then removed from the discharge compartment of the process water tank. The remaining water was then pumped out of the discharge compartment into the holding tank. The solids that had overflowed the barrels and settled were recovered by shovelling this material into the tailing barrels. Equal amounts were added to each barrel. A wet vacuum cleaner was used to recover the solids that could not be removed with a shovel. The final step in the clean-up was cleaning up the small amount of solids that spilled on the floor during the test run and subsequent clean-up.

Prior to remixing, all excess water was removed from the barrels containing the gravel. The initial tests, using the -1/4" samples, was retained in 12 to 14 barrels approximately 1/2 to 2/3 full. Later tests using the -3/4" material required 22 to 24 barrels. The gold

separated from the concentrate of the previous run was divided into portions of equal size. The number of portions corresponded to the number of barrels containing the recovered sample.

Remixing the gravel and gold was accomplished by the following procedure:

1. A tailing barrel containing solids recovered from the previous test was dumped into the small hopper that fed the cement mixer. The barrel and hopper were completely rinsed to ensure all solids were in the mixer. The mixer was tilted down and all free standing water was decanted into 5 gallon pails. The pails were allowed to stand for 3 minutes to settle out any solids, after which the fluid portion was poured into the discharge compartment. The mixer was started and tilted down until the gravel was almost overflowing. Maximum agitation and mixing were achieved at this setting of the mixer drum.
2. The correct proportion of gold was added at this point. The gold was slowly sprinkled into the agitating gravel at a point where maximum dispersion occurred. the mixture was allowed to mix until the solids appeared to be homogeneous with no clay lumps.
3. The mixer was stopped and the contents discharged into a sample barrel below the mixer. The barrels of reconstituted gravel were filled to within a few inches of the top. The tailing barrels containing the gravel prior to remixing were only partially full. The discharge of mixed gravel from the mixer was periodically interrupted due to the sample barrel under the mixer becoming full. After replacing the full barrel with an empty one the discharge of gravel was continued until the mixer was empty. The -1/4" test gravel filled nine barrels after remixing. Eighteen barrels were required for the -3/4" test samples when remixing.

3.4 Preparations for Test Run

Each test run required a number of procedures be completed prior to actual gravel sluicing:

1. The appropriate riffles and matting were installed in the sluice. The riffles were fabricated slightly narrower than the inside sluice width to facilitate easy installation and removal. To prevent gold washing through the sluice by travelling along the gap between the sluicewall and riffle, the riffle sections were tightly placed against alternate sides of the sluice. This meant that sections #1 and #3 were against one side of the sluice and #2 and #4 against the other side. The expanded metal was secured so that over the entire area covered by the riffles, the metal was just pressing into the matting where it made contact. This prevented underwashing and exposed the maximum riffle profile to the gravel slurry.
2. The slope of the sluicebox was set at the desired value. The two settings most often used were 1 5/8 in/ft (13.5 mm/m) and 2 3/8 in/ft (19.8 mm/m). These were the minimum and maximum values of slope the test facility could be operated at.
3. The holding tank was drained into the process water tank.
4. Make-up water was added to the process water tank to make up for any small amount of spillage from the prior test. The water required was usually minimal (< 50 US gallons).
5. Empty tailing barrels were placed in the discharge compartment to receive the sluiced gravel. The discharge-trough system was installed to feed the barrels.

6. As much water as possible was decanted by bailing from the barrels containing the feed sample. This prevented excessive splashing during discharge into the feed hopper and provided the best sample condition for obtaining a uniform rate of feed from the hopper. Too much water in the feed gravel caused some severe fluctuations in solid flow from the hopper.
7. The feed hopper was loaded at this time. The hopper could contain four full barrels of feed gravel at one time. A plywood panel was placed over the gate opening when the first barrel was dumped into the hopper. This prevented a surge of fluidized gravel from shooting out the gate opening when the barrel was dumped. The panel was removed after the first barrel was dumped. There was no discharge from the gate during subsequent loading as the gate was blocked by packed gravel from the first barrel.
8. The process water pump was started and the discharge manifold valves adjusted so that approximately 200 USGPM (2.5 m^3) were flowing through the sluice.
9. The very fine solid material that had settled in the suction compartment was agitated with shovels. This was continued until the process water was carrying a large amount of the fines ($> 10,000 \text{ ppm}$). Most of the fines remained in full suspension for the duration of the test. The process water, as it circulated, was similar in appearance to the effluent from a normal primary settling pond in the Yukon Territory.
10. The manometer lines and U-tubes were flushed with clean water. Clays did tend to obstruct the plastic tubing that connected the orifice plate pressure ports to the manometers.

11. The water flow was adjusted to give the selected head differential across the orifice plate. The water was coarsely adjusted by using the butterfly valve between the pump and orifice plate manifold. The exact settling required was then achieved by opening or closing the gate valve of the bypass line.

The speed of the belt conveyor under the feed hopper was found to increase slightly during the course of some early test runs. This increase in speed, up to 15%, was found to be due to the heating up of oil in the hydraulic system as a test run progressed. To prevent this change in feed rate, the belt was run for a period of approximately 15 minutes immediately prior to the loading of the hopper. This procedure was introduced prior to test run #22 and continued through the final test runs.

3.5 Sluicing

Sluicing commenced when gravel was first introduced to the entry flume. The water was already flowing at a pre-determined rate with as much suspended solids as possible. Each sluice run was timed with a digital stopwatch which was started when the first gravel dropped into the entry flume. Upon entry of the first gravel to the sluice, the following observations and adjustments were made.

1. The belt scraper was checked to ensure it was adequately cleaning the belt. It was positioned against the belt 6" from the bottom of the discharge pulley on the horizontal section of the belt. The force of the rubber scraper against the belt was adjusted to be just sufficient to clean the belt. Excess force was found to bind the belt and slow it down or in extreme cases, stop it completely. The scraper was checked at the start of gravel feed and after every readjustment of the slope of the entry flume.

2. The slope of the entry flume was set to minimize the flow disturbance at the sluice entry. Visual observation of the slurry behavior at the entry to the sluice was monitored to achieve this. The transition from the relatively smooth surface base of the entry flume to the riffled area of the sluice always caused some flow adjustment by the slurry. Minimizing the distance down the sluice that the slurry had to travel to attain equilibrium of depth and scour behind the riffles was considered adequate for the purposes of this program. The slope of the entry flume was quickly and easily changed using a small scissor jack. Most of these adjustments were completed within 15 seconds of starting the gravel feed.
3. The discharge splitter box was checked to ensure that no excessive spillage was occurring. A rubber skirt that formed the side of the box under the sluice had to be periodically adjusted to prevent spillage into the discharge compartment. High water flow rates, especially at the $2 \frac{3}{8}$ in/ft sluice slope, were the most difficult to contain.

Sluicing continued at the appropriate control settings until a process interrupt was encountered. Regular procedure required that sluicing be interrupted periodically to re-load the feed hopper or to replace the partially loaded barrels from the discharge compartment with empty ones (Figure 14). These interruptions were planned to accomplish both tasks at once, whenever possible. When the feed hopper was initially full, approximately one quarter of the full gravel load remained in it when the tailing barrels were all $1/2$ to $2/3$ full of solids. The tailing barrels were replaced and the hopper was loaded until full.

Care was taken to prevent unplanned periods of water flow without gravel feed. Known as "running clear", this condition scours out a portion of the concentrate volume that exists when the sluice is

processing gravel. It was not initially known whether this scouring mechanism would influence gold recovery and/or distribution. Planned intervals of water flow alone over loaded riffles were incorporated into five test runs.

The procedures developed to prevent "running clear" during shutdown and subsequent startup for a process interrupt were:

SHUTDOWN

1. Shut down pump.
2. Once water flow started to decrease visibly (about 2 sec elapsed time), immediately stop belt conveyor hydraulic pump.

STARTUP

1. Start water pump.
2. When first surge of water enters entry flume, gravel feed was started.

Proper shutdown left the riffles partially buried. The material transported during the period of rapidly declining water flow, was probably not concentrating the contained gold well. The duration of this solids transport was short (< 3 sec). The amount of gravel so treated was insignificant (<< 1%) compared to the total volume. Visual observations of properly executed shutdowns showed no scouring. The riffles operated at their normal scour depth until they filled up as the water flow decreased. During test #17, the final shutdown was poorly executed and significant movement of the gold was observed.

There were a number of unplanned interruptions. Some of these were foreseen with sufficient time to execute a proper shutdown. Poor gravel feed from the feed hopper gate could be caused by bridging or sticking. This condition could be detected with 1 to 5 seconds before the gravel flow to the sluice was disturbed. The few problems of this nature usually occurred during start-up. Particularly close observation of feed was adopted over this period to minimize such interruptions.

The most serious process interruption occurred when the feed hopper conveyor stopped suddenly without warning. This occurred twice during the early phase of the project. Both times the belt scraper had jammed against the belt tightly enough to overcome the torque supplied by the hydraulic motor. The belt scraper was modified and performed well afterwards.

Sluicing, with associated interruptions, was continued until all the barrels of prepared gravel had been processed through the sluice. The final shutdown involved shutting off the pump about 3 seconds before the gravel feed ended. Excessive concentrate volume would result if the water was shut off too soon. Conversely, there could be considerable scour if water flow continued after the gravel feed ended. Great care was taken to prevent scour where it was not desired. In some tests the concentrate recovered was considerably greater in volume than the instantaneous volume of solids in the operating sluice. It was considered preferable to bury the riffles by prematurely stopping the water flow rather than risk scouring by stopping it too late.

3.6 Cleanup and Gold Extraction

The concentrate contained in the riffles after final shutdown was drained of all but interstitial water. This was considered complete when the water draining from the sluice had declined to a rate of a few drops a second. The discharge splitter box was washed out and removed from under the sluice. A plastic tub was placed under the sluice discharge. The screws holding the riffles down were removed and cleaned off by brushing. The number four (#4) section of riffle, closest to the discharge, was then raised until the matting beneath it could be removed and placed in the tub under the sluice discharge. The riffle section was then rinsed off above the exposed sluice box base. Care was taken to not disturb the concentrate in the #3

section. The clean riffle section was removed. The material that remained on the sluice where the carpet was removed was washed over the discharge end of the sluice into the tub. The matting section, concentrate, and water were transferred to a labelled 5 gallon pail for further processing. The tub was returned to the sluice discharge and the process repeated for the number 3 section of riffles and matting. Section number 2 and then number 1 were cleaned in a similar manner. Usually, most of the gold was retained in section number 1 closest to the sluice entrance. The rubber base of the sluice had to be carefully washed after removal of each riffle and matting.

The matting sections were removed from the pails and separately rinsed. All solids washed from the matting were returned to the appropriate concentrate sample. The matting sections were placed in a plastic container for storage until needed again. Each concentrate sample was then screened separately initially at 18 mesh. The oversize was panned during the first few test runs to check for the presence of gold. This was subsequently discontinued, except on a periodic basis for checking. It became apparent that, when properly screened, virtually all the gold was contained in the minus 18 mesh fraction. Further treatment depended on the amount of concentrate remaining in each sample. Tests with expanded metal yielded up to five pounds (2.3 kg) of -18 mesh concentrate per section. The dredge riffles could yield as much as 30 pounds (13.6 kg) per section when screened. Two different extraction procedures were used to separate the placer gold from the remaining concentrate.

Expanded Metal-Concentrate Procedure

The four samples of concentrate, one from each section of the sluice, were separately screened at Tyler 32 mesh and Tyler 60 mesh. The three size fraction of each sample obtained were -18 + 32, -32 + 60, and -60 mesh.

The 32 mesh and 60 mesh screens were used because they were intermediate between the size fractions of seeded gold. By screening the sample first at 32 mesh all the -20 + 28 gold was retained in the oversize fraction. Equally important, the opening size would readily pass the finer gold, especially that close to 35 mesh in size. A similar argument justified using the 60 mesh screen to affect a separation of the -35 + 48 mesh gold from the -65 +100 mesh gold.

Twelve samples were thus readied for the final gold extraction. They consisted of three size fractions of concentrate (-18 + 32, -32 + 60, -60) for each of the four riffle sections.

The sized samples were then treated separately on the Goldhound spiral bowl concentrator. Each sample was treated twice. Liquid dishwashing detergent was used as a wetting agent and to reduce surface tension of the water used. This greatly reduced the tendency for gold, especially the -65 +100 mesh fraction, to "float" during processing and be transferred to the tailings.

The first treatment consisted of placing the entire size fraction sample (usually approx. two pounds) in the bowl of the concentrator. The recovered concentrate contained virtually all of the gold and some of the heavy minerals.

The volume of heavy minerals present depended on the shape of the gold, being recovered. The relatively flat -20 + 28 mesh gold fraction was the most difficult to separate on this type of concentrator. Large amounts (up to 20-30 gms) of heavy minerals were often recovered in the concentrate of the coarse samples. The tailings from the first treatment were carefully checked visually and retained for remixing with the gravel sample. The concentrate from the second pass consisted of placer gold particles with few impurities. The amount of impurities depended on the amount of gold recovered. The samples from the #1 section of the sluice generally

contained from 30 to 70 grams of gold in each size fraction. The amount of impurities was estimated to be less than 5% by volume. The samples from the last riffle section frequently contained less than a gram of gold. The amount of impurities, with gold samples this small, was often larger than the amount of gold recovered.

The Goldhound worked well for concentrating the gold contained in each sample to a point where panning could rapidly yield a pure product. Samples were processed in the same sequence for each test run. The coarse fraction (-18 + 32#) from the #1 section was processed first. The concentrator was adjusted while the large amount of gold in the sample (usually greater than 50 gms) was removed. The sample was then retreated at the same bowl angle but with a higher water flow. The same bowl angle was used to treat the rest of the coarse samples in the sequence; #2, #3, and finally #4 section.

The intermediate fraction (-32 + 60 mesh) of the #1 section was treated next. The bowl angle was adjusted to optimize separation at the smaller particle size. The remaining intermediate sized samples were then treated in the same order as before.

The fine fraction (-60 mesh) was processed similarly after adjusting the bowl angle.

Each gold concentrate sample was then placed in a 15" (38 cm) plastic gold pan. A hand magnet was used to remove all the strongly magnetic minerals, mostly magnetite. The remaining impurities consisted mainly of hematite and minor amounts of scheelite. These impurities could be washed away from the gold by careful panning. A suction bottle was used to recover gold from the areas of the pan where the impurities had been removed. The material remaining was repanned and more gold recovered. This was continued until there was only a few particles of gold, (never more than 10 mg), remaining in the pan.

The gold recovered was placed in clean beakers and rinsed with hot clean water 3 times to remove any soap residue. The tendency of the gold to "float" in clean water required that the final rinsing steps be done very carefully. The 12 beakers of gold were placed in an oven at 275° until the water had completely evaporated. After cooling the gold was weighed to an accuracy of .01 gms on a digital top loading balance. The impurities present in the dried samples were minor. In the large gold samples (more than 5 grams) the error was certainly much less than 1%. The weight of impurities in the smaller gold samples introduced larger errors but, in any event, would rarely have exceeded 1%.

Periodically, the recovered gold had small amounts of mercury coating some of the particles. Mercury had previously been used in the manometers and a small volume was lost into the orifice plate manifold. The early tests using high water flows flushed some of the mercury into the sluice where it adhered to the gold. When mercury was observed in any sample, all were washed briefly in nitric acid before rinsing and drying.

Dredge Riffle Concentrate Procedure

The large volume of concentrate recovered from each dredge riffle section could not be conveniently treated on a batch basis. Screening these large samples at 32 mesh and especially 60 mesh was very time consuming and tedious. An alternate procedure was developed.

A Syntron vibratory feeder with a small conical hopper (Figure 13) was used to feed the -18 mesh sample into the Goldhound concentrator. The feeder was adjusted to provide a feed rate of approximately 500 gms per minute. The concentrate from section #1 was processed so that most of the heavy minerals (S.G. >5) were recovered. The processing continued until the heavy minerals passing into the concentrate ceased to contain gold particles. The concentrate and tailings were removed from the spiral concentrator. The gravel

recovered in section #2 was processed next, followed by #3. The material recovered in the small #4 dredge riffle section was also treated at this time.

The four concentrated samples, one from each dredge riffle section, recovered from the Goldhound were then hand sieved at 32 and 60 mesh. The resulting twelve samples were processed further in a manner identical to that employed for the concentrate recovered from the first Goldhound treatment of the expanded metal samples.

4.0 Results

A concise summary of data gathered from twenty five pilot-sluicing tests is presented in Table 1. This table lists results of individual tests numbered 4 to 29 in their chronological sequence. The data for each test run is presented in the Appendix. Each data sheet contains tabulations used for analysis along with appropriate statements of objectives and comments. Tests numbered 1 to 3 were preliminary to remove the "natural" placer gold present in the -1/4 dry-screened fraction of the Sulphur Creek gravel and to develop suitable operating procedures. Results from these preliminary tests are reported later. Test 15 also not included in Table 1, was to recover gold from the -3/4" wet screened gravel added later.

Tests 4 through 9 were conducted using solid feed rates representative of operating placer mines in combination with water flow rates, riffle type, and slopes considered most effective by the authors. The results demonstrated the high efficiency of sluicing for recovery of gold as fine as 100 mesh. Test 10 was conducted under conditions similar to Test 7 but with 30 second periods of solid feed stoppage for every 2 minutes of normal solid feed. The purpose of this test was to investigate the effect of sudden stoppages in solid feed to an operating sluice. Tests 11, 12, and 13 were identical to Test 9 except for different solid feed rates. This demonstrated the effect of variation in solid feed rate on gold recovery when all other

conditions were held identical. Test 14 was a repeat of Test 6 except that "clear" (no solid feed) water was pumped to the sluice for 2 hours after all solids had been processed. The common practice, in the Yukon of pumping water through a sluice without ore feed for long periods was the basis for this test procedure.

Tests 16 through 20 used dredge riffles (1-1/4" angle iron) in combination with variable settings similar to the previous expanded metal tests (#4 through 14). Test 21 was conducted using expanded metal with a combination of other settings that were not previously used but thought to be effective. The purpose of this was twofold, first, the acquisition of another data set using 400 USGPM water flow, 2 3/8" in/ft. slope, and expanded metal. The second purpose was to process the entire feed using known efficient recovery conditions to determine whether the significant amounts of gold not recovered during the dredge riffle tests was due to poor recovery or gold losses from the test facility. Test 22 was conducted using expanded metal in an orientation reverse to its normal emplacement. Test 23 used cocoa matting as the matting under expanded metal to determine its effectiveness when exposed to severe scour compared to Nomad matting. The conditions employed were identical to test #10 where the effectiveness of Nomad matting exposed to severe scour was investigated.

During test 24 through 27 the feed gravel was - 3/4" due to the inclusion of the - 3/4" + 1/4" fraction of the original gravel sample in the test gravel. Conditions selected during these tests were chosen to determine the effects of the larger particles on gold recovery. A variety of conditions representative of the previous series of tests, using the -1/4" fraction were employed.

Test 28 was a scavenger run, conducted at variable settings known to be efficient, to remove virtually all the gold remaining in the test gravel at the inclusion of Test 27. Test 29 used only 4 barrels

of feed gravel seeded with -100+150 mesh gold to determine the recovery of this size fraction under conditions similar to those of the very efficient Test 4.

Results in Table 1 show gold weight distributions in the various sluice sections for each size fraction seeded into the feed. Over the course of the entire series of pilot-scale test runs, nearly 13 grams out of 275 grams total gold added was lost due to accidental spillages, entrapment in equipment or other unknown causes. In order to calculate total gold recoveries during each test, this loss has been apportioned out equally to each test.

Every data set is comprised of the weight, in grams, of gold recovered for each size fraction from the test riffles sections. The tests using expanded metal as a riffle had 4-2' long test sections. Each section yielded 3 gold samples of different size. A total of 12 samples of gold were recovered for weighing during each of these tests.

The tests with dredge riffles had only 3, 2' long, test sections. The last section (#4) was a short riffle section used to prevent unusual conditions from occurring at the end of section #3. The gold from this riffle section was usually recovered but not included in the data set. For this reason the data sets for the dredge riffle tests have only 9 samples, three from each of three test sections.

The time consuming, complicated procedures used to gather these data certainly introduced some error. Fluctuations in the solid feed rate and process interrupts were considered to be the largest source of error in the resulting data sets. The extraction procedures, particularly sieving, could also lead to considerable errors if not carefully done. The data from test run #4 were particularly susceptible in this regard. To prevent errors, screening was performed very carefully and constantly checked.

It became apparent that the data sets generated showed many similar characteristics even though the actual weights of gold recovered varied substantially. The similarities noted, in order of their perceived importance, were:

1. The largest proportion of the gold recovered from each size fraction was always in section #1, at the feed end of the sluicebox.
2. Section #1 always recovered a higher proportion of the coarse fraction of gold than of the fine fraction.
3. Conversely there was always a higher proportionate amount of fine gold than coarse gold in sections 3 and/or 4 of the sluice.
4. The total amount of gold recovered in section #1 was much greater than that of section #4 for all the test runs with expanded metal. The usual amount recovered in section #4 comprised less than 2% of the total gold recovered. The highest proportion recovered in section #4 for the expanded metal runs was 8% for test run #13. With the exception of test run #17, the dredge riffles tests showed a similar relationship between sections #1 and #3. However section #3 contained a much higher proportion (from 10-20%) of the total gold recovered when using dredge riffles.

The above observations are consistent with comments received from operators regarding the distributions of gold in their sluiceboxes. This would be expected of a representative pilot-scale test and gives some assurance that the data gathered for this report reflects the behavior of full scale sluice (albeit, under very carefully controlled conditions with an artificially enriched and sized gold content).

5.0 Discussion

5.1 General Discussion

One of the unique aspects of this project was the ability to observe the behavior of the slurry within the riffles and immediately adjacent to the plexiglass sidewalls. The turbulent eddies that formed around the riffles as a result of the slurry flow were easy to distinguish. A video tape was made to record many observed flow patterns. Close observations of the concentrate bed showed the presence of several different zones within the sluice. A dispersed shearing bed of mineral particles was observed immediately adjacent to the high velocity mixture of water and solids in the eddies. The size and geometry of these eddies appeared to be most significant for correlation with the noted recoveries. The eddies could be characterized by the maximum depth of scour between succeeding riffles. This was expressed as a visual observation of the maximum proportion of the depth between a line intersecting the tops of the riffles and the top of the matting where scour existed. Within seconds of starting each sluice test a stable pattern of scouring was established behind the riffles. The size and geometry of the turbulent eddies and their associated scour pattern did not change if the variables were maintained at a constant value. The scour depth behind the initial riffles (feed end of section #1) was usually different from the uniform scour that existed throughout the rest of the sluice. This was due to unavoidable flow adjustments that inevitably occurred in the transition from flow over the smooth entry flume to the riffled sluice. There existed, for each riffle type, a characteristic scour pattern for areas in the sluice where scour depth was equal. The scour depth, which was directly proportional to the size of the eddy responsible for it, varied, depending on the operating variables used for each test run.

It was observed that, except for one or two riffles at the sluice entrance, an energy equilibrium existed throughout the sluice. The energy gained due to the sluicibox gradient appeared to be dissipated by fluid resistance in two ways:

1. resistance against the flow by the particles of aggregate in the slurry
2. frictional losses due to flow over the riffles, concentrate and matting, where exposed.

The slurry flow appeared to be distinguishable as three phases under steady state conditions during gravel feed as shown in Fig 15.

Phase I - This was considered to be the volume occupied by the turbulent eddies around the riffles. Flow in this region was highly rotational about a horizontal axis that coincided with the center of the eddy cross section. Establishing the boundaries of this phase was somewhat arbitrary. In general, any volume below a plane touching the top edges of the riffles occupied by solids and fluid in motion was considered to be in Phase I flow.

Phase II - The slurry immediately above the riffles was observed to flow in a plane roughly parallel to the sluice base. This phase contained a high proportion of solids and represented the volume of slurry where mass transport of most of the solids through the sluice occurred. Particulate solids were continuously being exchanged between the volumes occupied by Phase I and Phase II flow. The upper boundary of this phase was considered to be where solids concentration dropped to a relatively low value. In most tests this phase II appeared to be approximately 1/2" to 1" (12 to 25 cm) thick.

Phase III - The remaining volume of slurry above Phase II flow was considered in Phase III flow. It was characterized by a low solids concentration and the solids contained were mostly silt and clay sized particles.

The active volume occupied by Phase I flow appeared proportional to the velocity of the slurry at the boundary between Phase I and Phase II. The larger the velocity at this boundary the larger were the eddies that formed between the riffles. The energy required to

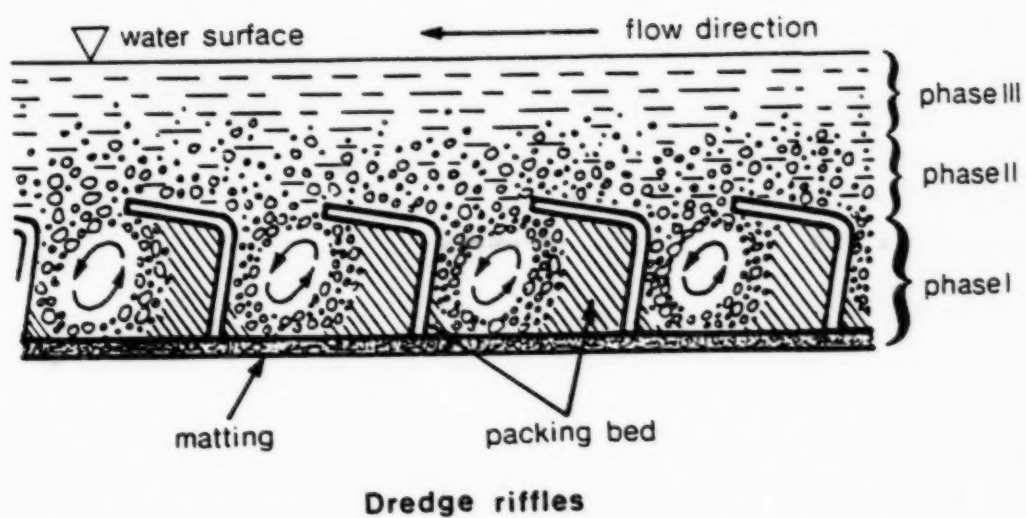
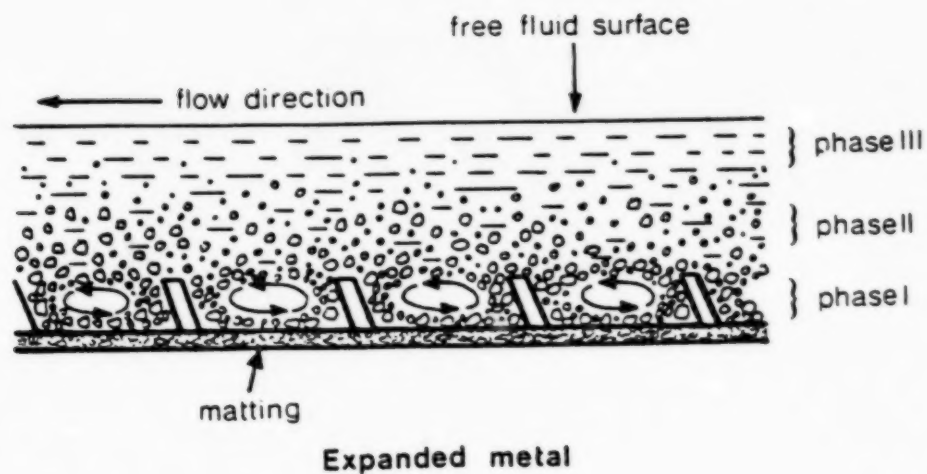


FIGURE 15

sustain the rotational flow was provided by shear transfer from the base of the Phase II flow above. High velocities at the base of phase II transferred enough energy to the phase I region to sustain large eddies. The upper limit of eddy size was dictated by the physical boundaries created by the riffles and matting. Low velocities could only transfer enough energy to sustain small eddies (shallow scour depth). As the velocity at the base of phase II approached zero the volume occupied by the eddies became progressively smaller. Phase I flow did not exist when the slurry velocity at this boundary was so low that the energy required for the smallest eddies was not available. This corresponds to the situation where the riffles become "buried" in an overloaded sluiceway.

Simply stated, the energy available in excess of that required to move the solids along the sluice appeared to be dissipated by the turbulent eddies. The energy was consumed by transporting the solids in the eddy at any instant and frictional losses, mostly to the dispersed shearing particle bed that formed in the uppermost layer of concentrate.

With only water flowing, the gravel was scoured out until the turbulent eddies reached the upper size limit of their development. In this condition the gravel remaining was located in corners formed by the riffles and matting and in the matting itself. It was noted that clear water (no gravel feed) scoured out the riffles completely except at very low water flow rates (< 50 USGPM).

The range of scour patterns that could occur were observed by setting the process water flow rate at approximately 150 USGPM (1.85 m^3) and introducing gravel at a steadily increasing feed rate. A small volume of gravel was first introduced to enable the riffles to fill with gravel. Gravel was next introduced initially at a slow feed rate (50 lbs/min). The feed rate was slowly increased until the scour depth began to decrease as the eddy size started to shrink. The feed rate at which this condition occurred depended on the water flow and

riffle type. As the gravel feed rate was increased from this value, the turbulent eddies shrank and packed solids progressively filled more of the space between the riffles. Eventually, the point was reached where eddying ceased and the riffles were almost totally buried.

Subsequent discussion of each control variable, and how it influenced gold recovery, will rely heavily on a description of how changes in these variables affected the scour patterns in the sluice.

5.2 Riffles

The type of riffle, employed had a great influence on the behavior of the gravel/water slurry. Each of the two riffle types studied showed a characteristic pattern in the resulting data sets.

The most significant features of riffles that rely on horizontally oriented turbulent eddies for concentration are:

1. The effective distance above the matting surface the riffles project. For the expanded metal this was $1/4$ of an inch (6.35 mm). The angle iron was $1\ 3/8$ " (35 mm) in height.
2. The spacing between the riffles. This was measured as the distance from the downstream edge of one riffle to the upstream edge of the succeeding riffle. The expanded metal was $3/4$ " (19 mm) at the widest section of the opening. The dredge riffle spacing was $1\ 1/4$ ".

Expanded metal was very effective in recovering the seeded placer gold. Except for test run #13, the recovery of the -20+28 mesh gold was always in excess of 95% using expanded metal. The extremely high solid feed rate during test #13 (1260 lb/min.ft. or 570 kg/min) was considered the reason this particular recovery dropped below 95% in the coarse size fraction.

Recovery of the -35 + 48 mesh gold was slightly poorer, but usually in excess of 95%. During test run #13 this dropped to the order of 90%.

Losses of fine gold (-65+100#) were higher, but recovery was still good. At the higher flow rates of water and solids, less than 90% of this fraction was retained in the sluice. The proportion of this gold found in the #3 and #4 section was considerably greater than the coarser gold populations, as was expected. The worst recovery of fine gold using expanded metal was 75% during test run #13.

Very similar scour patterns developed in the sluice during the most effective test runs. The scour depth was from 1/2 to 3/4 of the maximum possible. The eddies were elliptical in cross section extending from one riffle to the next on its long dimension. The short dimension was equal to the scour depth (see Figure 12).

It appeared that for any particular feed rate of solids, a limited range of process water flow rates gave this scour pattern. Increasing the slope would reduce both the upper and lower boundaries of this range of flow rates. Reducing the slope would have an opposite effect. In an environment where more energy was available for eddy formation (for example a high water flowrate, steep slope, low solid feed rate) the scour would increase, exposing some or most of the matting. Recoveries, using expanded metal, under these conditions were still generally high (>90%) for even the fine gold.

Scour depths less than 1/2 of the maximum depth resulted in lower gold recoveries. The relatively low recovery of test #13 resulted from an observed scour depth from 1/3 to 1/2 of the maximum.

The eddies that formed when using expanded metal reversed to its normal orientation (test #22) were very similar to those in its normal configuration under otherwise identical conditions. Less concentrate weight was retained due to the orientation of the riffles. This

influenced particularly the location of the coarse gold which spread further down the sluice. Recovery of the -35 mesh gold fractions appeared to be very similar (test #22 vs. #8 and #4) for either riffle orientation.

Dredge riffles were not as effective as the expanded metal under similar operating conditions. Comparison of test #9 with #18, test #7 with #16, and test #21 with #20 showed that the gold had a greater tendency to be transported through riffle section #1 for the dredge riffles. The overall recovery of the first 3 sections for these tests was much greater for the expanded metal riffle system. For the -20+28 mesh fraction, recoveries between 75 and 90% were documented using dredge riffles compared to recoveries in excess of 95% using expanded metal. Between 70 and 90% of the -35+48 mesh gold was retained using dredge riffles. From 50 to 90% of the fine gold present was found in the dredge riffle concentrate. The amounts of gold that passed far enough through the sluice to be caught in sections #2 and #3 was considerable as would be expected with these lower recoveries.

Examination of data obtained during tests operating under various scour conditions showed some unusual trends. During test #16, which was the most effective dredge riffle test, the riffles were observed to be scouring to approximately 1/4 to 1/3 the maximum. The overall recovery and distribution down the sluice was remarkably similar for each of the three gold size fractions. Between 85 and 90% of each size fraction was recovered from the 6' test section. This was unusual, because coarse gold was always recovered more efficiently than the fine fraction for all tests with expanded metal.

When scour depth increased to 1/2 the maximum, (as in test #18) recovery of the coarse gold fraction dropped to around 80% (from 85-90% value above) and recovery of the fine fraction dropped even farther to - 65%. At conditions of maximum scour in the dredge riffles (test #20) recovery of the coarse gold improved to around 90% but fine gold recovery continued to deteriorate to - 50%.

The unusual results of test #17 were due to a poorly executed shutdown, which visibly moved gold and concentrate. Test 18 is a repeat of test 17 to determine recoveries with a proper shutdown.

Although no detailed explanation can be offered for the above behaviour at this time, optimal scour depth appeared to be from 1/4 to 1/3 riffle height when using dredge riffles.

5.3 Solid Feed Rate

By suitably adjusting the water flow rate, similar scour conditions could be maintained for a wide range of solid feed rate settings. Even under these roughly equivalent flow regimes, the higher the solid feed rate, the lower was the recovery of each size fraction. Consistent with this observation was the tendency for gold to travel further down the sluice as the feed rate became greater. Shorter particle residence times in the sluice and higher collision frequencies between gold and gangue particles at higher solid feed rates were thought to be responsible for this phenomenon.

Figure 16 presents the recovery data from 6 tests using expanded metal where all variables except solid feed rate were identical. The range of solid feed rates tested would correspond to full scale commercial sluicing treating from 3.6 yd³/hr to 50 yd³/hr of bank run gravel (50% of which is screened off or otherwise removed prior to sluicing) for each foot of sluice undercurrent or side run width.

Decline in recovery from each size fraction was relatively small as the solid flow rate increased over the range of values at which a particular scour pattern could be sustained. By this reasoning, the use of high solid feed rates should not, in itself, be responsible for high gold losses unless such feed rates result in inefficient scour conditions. Practically speaking, the sluicebox configuration and capacity of water pumps might limit the solid feed rate to a value above which optimal scour conditions cannot be obtained.

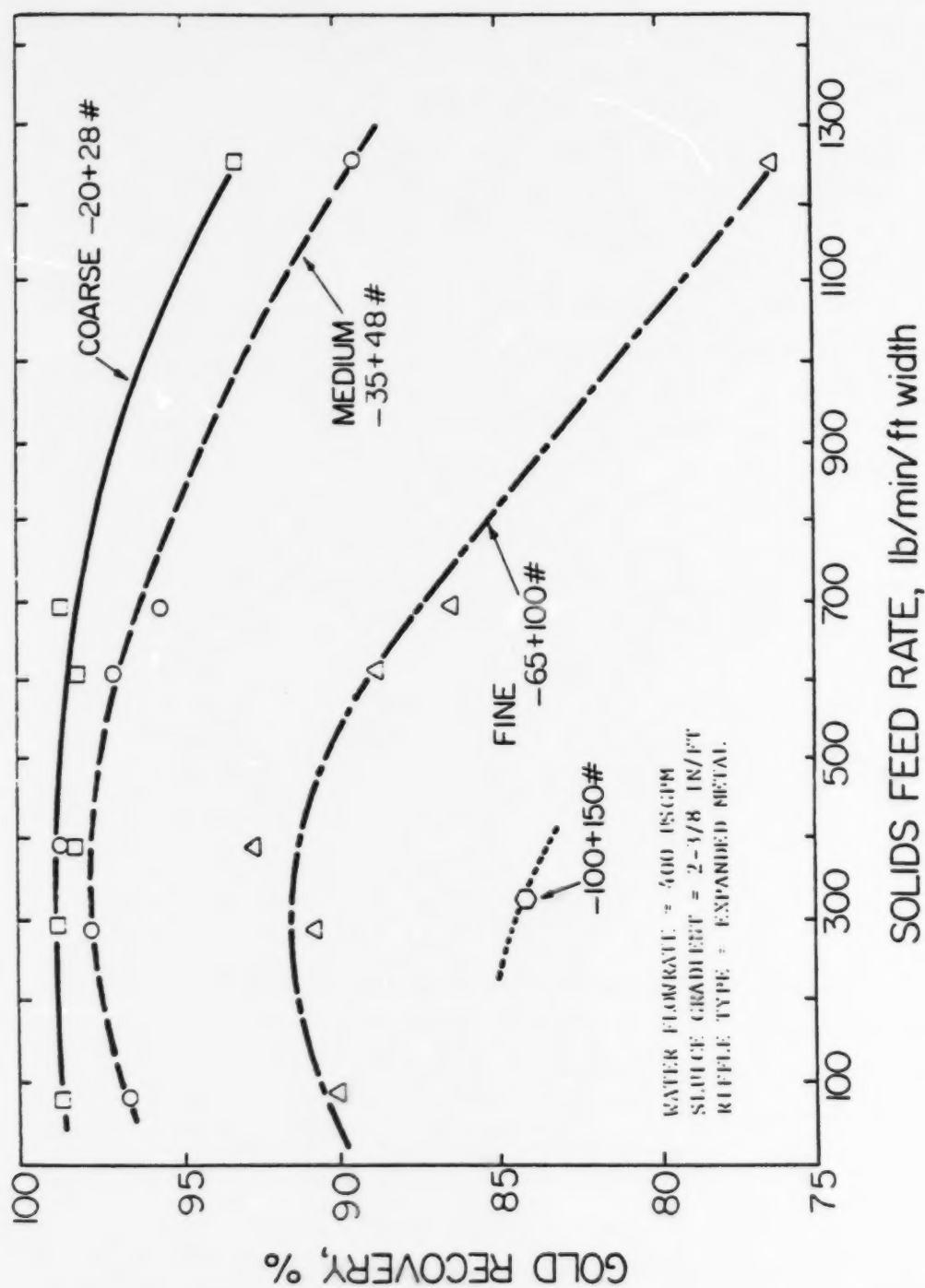


FIGURE 16 Sluice Performance Vs Solids Feed Rate

5.4 Water Flow Rate

The effect of changing water flow rate on gold recovery, when all other conditions remained constant, depended on the scour conditions present. The best gold recovery occurred when the flow rate produced the most efficient scour pattern for that riffle type. Flow rates less than this produced less scour and poorer recovery. Higher flow rates resulted in more scour with the attendant reduction in recovery.

The range of values over which the flow produced favourable scour conditions was reasonably large. For example when using the (1-10H) expanded metal riffles very good recovery was obtained for test runs #4 and #6. The flow rate in test run #4 was 160 USGPM ($1.98 \text{ m}^3/\text{min}$) approximately 1/2 that in #6, 290 USGPM ($3.6 \text{ m}^3/\text{min}$). Comparison of results from test runs #7 and #21, conducted at a higher slope, showed a slightly larger drop in recovery when the flow was increased from 290 USGPM to 400 USGPM ($4.95 \text{ m}^3/\text{min}$). Gold recovery at the higher flow rate of 400 USGPM was still very high (90% for -65+100 #Au).

Similar flow rate comparisons for dredge riffles in test runs #16 and #20 showed a greater effect on gold recovery. The overall recovery was about 10% lower at the high flow rate (400 USGPM). However, the recovery of coarse gold was noticeably improved at higher flow rates. Recovery of fine gold was much lower during the high flow rate (70% vs 90%) resulting in an overall reduction of gold recovery.

High flow rates associated with maximum scour produced lower recoveries than optimal. The riffle type was very important to the recovery under these conditions. Recoveries of the seeded gold using expanded metal were still high (see test run #11) at these high flow rates. Fine gold recoveries were relatively poor (<70%) at maximum scour.

Relatively low water flow rates tended to "bury" the riffles and if severe enough, all rotational or Phase I flow ceased. Recovery under this condition was not investigated quantitatively but was assumed to be relatively poor.

5.5 Sluice Gradient

The sluice gradient affected the scour conditions by determining the energy gradient of the flow. Suitable setting of water and solid flow rates could produce any desired scour condition. As the slope increased, the amount of water required to process a specified gravel feed rate at the same scour pattern decreased. When all conditions, including water flow rate, were maintained at constant settings, an increase in slope caused an increase in scour. A decrease in slope, or flattening, produced less scour. Gold recoveries were consistent with the scour patterns exhibited.

The greater the sluice gradient the higher the solids transporting capacity of the water. Over the range of values tested ($1 \frac{5}{8}$ - $2 \frac{3}{8}$ "/ft) changes in sluice gradient, did not significantly affect gold recovery if similar scour conditions were maintained.

5.6 Surging of Solid Feed Rate

The effect of surging was investigated during test runs #10, #19, #23, #25, and #27. These tests confirm that surging reduced gold recovery but the effect was only slight. As also expected, surging resulted in gold being distributed further down the sluice.

Prior to these tests we expected that sudden, periodic interruptions in the gravel feed would be highly detrimental to gold recovery. That this was not the case could only be attributed to the ability of the gold to remain well trapped in the matting during the period when the "clear" water flow scoured out the riffles. This scouring occurred very rapidly (< 1 sec) when the gravel feed was stopped suddenly. At maximum scour the matting was usually mostly exposed and much of the gold recovered was buried within the matrix of the matting. It was obvious that the recovery under conditions of maximum scour was strongly influenced by the matting type, as discussed in the next section.

5.7 Matting

The matting type influenced the gold recovery when scour conditions exposed it to slurry flow. The matting employed in most of our testwork (Nomad matting) was very effective in retaining gold when exposed. The open weave and flexible rubber strands comprising this material allowed a relatively sheltered environment to exist where a dispersed particle bed could form in the matting. Comparison between Nomad matting in test #10 with Cocoa matting in test #23 showed that, although both were effective (> 90% fine gold retention), the Nomad matting was slightly superior as far as recovery was concerned.

When operating under conditions where scour was not allowed to become severe enough to expose the matting, the choice of matting is not expected to affect gold recovery significantly.

5.8 Upper Feed Size

To our surprise, there was very little difference in test results obtained at identical conditions using either minus 1/4" gravel or minus 3/4" gravel. Comparison of test runs #20 with #24, #19 with #25, and #6 with #26 show very similar recoveries and distributions for all three gold size fractions. The scour conditions for these comparison pairs were also similar as would now be expected for test runs recording with similar recoveries.

5.9 Test with -100 + 150 Mesh Gold

The final test run (#29) involved mixing 25.38 grams of -100 +150 mesh placer gold to the gravel sample. The unseeded gravel had been sluiced under efficient conditions during test #28 to remove as much gold as possible from the sample. Data from tests #4 and #5 showed that operating with expanded metal at a slope of 1 5/8 or 2 inches per foot with 160 USGPM of water and 325 lbs/min (150 kg/min) of gravel was very efficient (> 90% recovery) for processing the -20 +100 mesh gold.

The same highly efficient operating conditions were employed during test run #29 to determine the sluice recovery of -100 +150 mesh gold. Approximately 85% of this fine gold was recovered in the 8' pilot-scale sluice using expanded metal riffles. This is much higher than expected based on previously published literature(1). However, the amount of this fine gold recovered in riffle section #4 (0.38 gms) compared to that not recovered (4.08 gms) indicates that the probability of recovery of such fine gold is very low near the discharge end of the sluice.

5.10 Additional Observations

5.10.1 Unrecovered Gold

The weight of gold not recovered from each size fraction was usually much larger than the amount recovered in the last 2 foot riffle section. For example, in test #27, 0.12 gms of -20+28# Au was recovered in the test section #4 of expanded metal. The tailing from this same run comprised the feed for test #28. In test #28, a further 1.43 gms in total of -20+28 # gold were then recovered. No satisfying explanation for the relatively poor efficiency of the last riffle sections can be offered at present. It was evident that the probability of gold recovery in sluice section #1 was very high in some tests for all size fractions of gold. Comparison of the gold recovered in section #4 (or #3 for the dredge riffles) with that lost showed a marked decrease in the probability of recovery at the discharge end of the sluice. Comparison of the gold available at the beginning of each test section with the gold actually recovered showed a marked decrease in the probability of recovery of gold particles from section #1 to the last test section.

One factor contributing to this behaviour is that the gold initially enters the sluice travelling, for the most part, along the base of the entry flume. Particle sorting in the entry flume was such that most dense minerals quickly segregated to form a moving bed immediately above its base. It appeared probable that the particles

moving along the sluice immediately above the plane separating phase I and phase II flow would be the most likely to enter the rotational Phase I flow where they might be retained as concentrate. At the entrance to the sluice most or all of the gold was in this part of the flow. Mass exchange between the phase I eddies and phase II flow above is believed to increase the probability of gold particles moving upwards from the base of the nonrotational flow. As the slurry travelled through the sluice the distribution of gold in the slurry changed. A higher proportion was contained in phase II flow above the point where it could be captured in an eddy. Shape could also have played a significant role in this phenomenon and will be reported in more detail in the M.A.Sc. thesis to result from this study.

5.10.2 Suspended Solids

Prior to feeding any gravel during a test run, the fine particles in the "suction compartment" of the water tank were agitated to suspend as much fine material as possible. When combined with fines contained in the gravel, the resultant concentration of suspended solids in the circulating process water was of the order of 10,000 ppm. This test program, was thereby conducted at suspended solids concentrations that are considered representative of many Yukon sluice operations. Our results indicate that high gold recoveries are possible with high concentrations of suspended solids. The effect of suspended solids on sluice recovery has also recently been investigated by others.(4) This Alaskan study also concluded that high suspended solids contents (< 20% solids) in the sluice process water had negligible influence on gold recovery.

5.10.3 Gold Concentration

The gravel sample for this program was purposely very rich in comparison to normal placer mining ores. Our objective was to improve both sampling and assaying accuracies. The -1/4 inch test sample

contained approximately 80 ppm Au weight when prepared for testing. The -3/4 test sample was about 40 ppm in Au due to its being double the weight. Modern placer mines typically process material that averages between 0.3 and 2 ppm Au by weight. The richest ores that occur in small pockets along paystreaks can be much higher in gold content. The gravel feed to a typical sluicebox often varies between 0 ppm and 5 ppm Au. In some operations this can occasionally increase to over 10 ppm for short intervals.

The concentration of gold in the test sample, though higher than normal ores, was still very small in comparison to the concentration of gangue minerals. It is assumed the higher concentrations would produce data comparable to that for lower grade ores. Comparison of our lower grade test runs 24 through 26 with equivalent higher grade test runs performed earlier on the -1/4" gravel showed no difference in results even though both the upper feed size and feed grade had changed.

5.10.4 Packing of Riffles

The short duration of each test did not allow any kind of cohesion to develop in the concentrate section of the riffles from infilling of interstitial voids with clay. The effects of surging the feed and the use of different mattings were therefore not influenced by "packing". This condition certainly does occur in operating sluices which may operate over 50 hours between cleanups. Packing occurs in riffle areas where the solids are not being disturbed by eddying conditions. Any reduction or interruption of gravel feed will increase scour and can then remove all unconsolidated material, including gold, from the concentrate volume. This condition could obviously increase gold losses and should be prevented. More frequent cleanups and external agitation of the concentrate volume are two methods considered effective.

The presence of sharp edged angular aggregate in the feed might also cause packing. This frequently occurs when processing fractured competent bedrock or angular alluvium. These types of aggregate particle tend to lock together in the sluices and resist scouring. The net result is that very little active concentrate is retained. This is expected to decrease the probability of gold retention especially when extreme scouring occurs. Angle iron riffles seem especially prone to this packing condition. Pulsating riffles are used to overcome this problem. Information on the design and operation of pulsating riffles is available through the Klondike Placer Miners Association.

5.10.5 Sluicing Conditions

Flow conditions in the entry flume, in the transition to the sluicebox, and in the sluicebox itself were considered important factors in sluice performance. The design employed allowed the slurry to enter the sluice with minimum disturbance. The solid particles were well dispersed or liberated from each other and partial sorting, (segratation of high density minerals toward the base of the entry flume), had already occurred. There were no constrictions or obstructions to the slurry flow in the system. The gravel was fed as uniformly as possible into the entry flume over its entire width to distribute the gravel and gold evenly across the sluice. Sidewall effects were minimum and confined to two small standing waves generated where the rubber lining of the entry flume was fastened inside the sluice.

5.10.6 Particle Liberation

The material used during these tests was well broken up and dispersed. Possible practical problems of fine gold adhering to coarser aggregate were not observed. This is not the case in many sluicing operations. This can obviously drastically reduce the gold recovery.

Significant gold did exist in the original Teck placer gravel bound up in fines adhering to coarse aggregate. This was convincingly demonstrated during the later gravel preparation phase of this program. The original screening of the Teck gravel sample at 1/4" was performed dry. The screening appeared to work well and the only -1/4" material passing into the screen oversize was that adhering to the coarse aggregate. It was estimated that in excess of 90% of the fines were recovered in the screen undersize. Preliminary sluice treatment of this -1/4" dry-screened fraction (weighing - 7500 lbs) yielded 4.18 gms of placer gold. The -1/4" dry-screen oversize was subsequently wet screened at 3/4" resulting in addition of - 8500 lbs of -3/4" material. At this point all the -1/4" fines that had adhered to the coarse aggregate were recovered in the screen undersize. Subsequent sluice treatment of this sample yielded an additional 2.20 gms of "natural" placer gold. These results indicate that 34% of the total natural placer gold recovered from the Teck Sulphur Creek gravel was contained in the "fines" adhering to the coarse (+1/4") oversize from the original dry-screening preparation.

6. Conclusions

Visual observations during test runs and analysis of the data have produced the following conclusions:

1. Sluiceboxes, operating at waterflow rates and solid feed rates typical of the Yukon placer mining industry (from 100-700 lb solids/min/ft sluice width) can be an effective method of recovering gold as fine as 150 mesh.
2. Expanded metal riffles (such as the 1-10H) are superior to 1 1/4" dredge riffles for recovering placer gold between 20 and 100 mesh. Test runs using expanded metal were significantly more effective in recovering gold, especially the -65+100 # fraction than equivalent tests using dredge

riffles, as shown in Figures 17 and 18. The fact that expanded metal recovered more gold in less concentrate weight (typically less than 1/8 that in equivalent dredge riffle tests) strengthens the argument for its superiority.

3. The orientation of expanded metal riffles is not important to gold recovery. It would appear that the normal orientation (see Figure 12) is slightly better at recovering -20+28 mesh gold due to the larger volume capacity for active concentrate within the riffles themselves.
4. The practice of running "clean" or allowing the gravel feed to stop while water is flowing need not greatly affect recovery. More gold will be carried further down the sluice and losses can be slightly higher when this occurs. However, when riffles become packed, running clean probably results in much higher losses.
5. Contrary to other studies of sluicing, (1), our results indicate that coarsening the upper size of gravel in the feed from 1/4" to 3/4" does not significantly influence recovery. Our results indicate that treatment of material as coarse as 3/4" should not be detrimental to recovery of gold down to 100 mesh.
6. The scour condition that exists in the sluice is the most significant factor in predicting recovery. Each riffle type has a characteristic scour condition where gold recovery is optimal.
7. Normal variations in the solid feed rate can be tolerated in sluicing without excessive gold losses. High feed rates that overload the sluice will bury the riffles and recovery is expected to suffer significantly. It is possible that periodic gradual reductions in feed rate would assist the

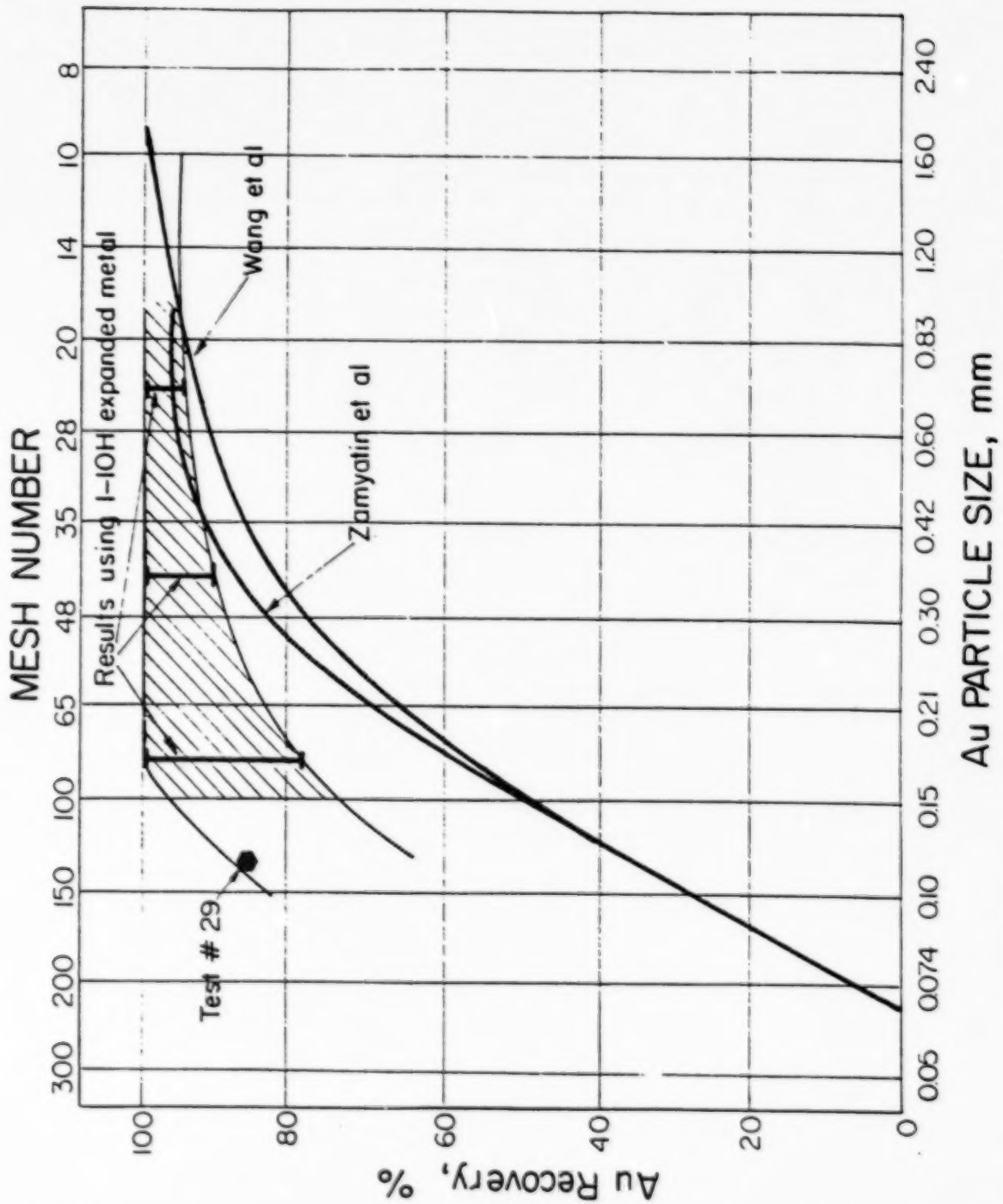


FIGURE 17 Gold Recovery Versus Particle Size Using Expanded Metal

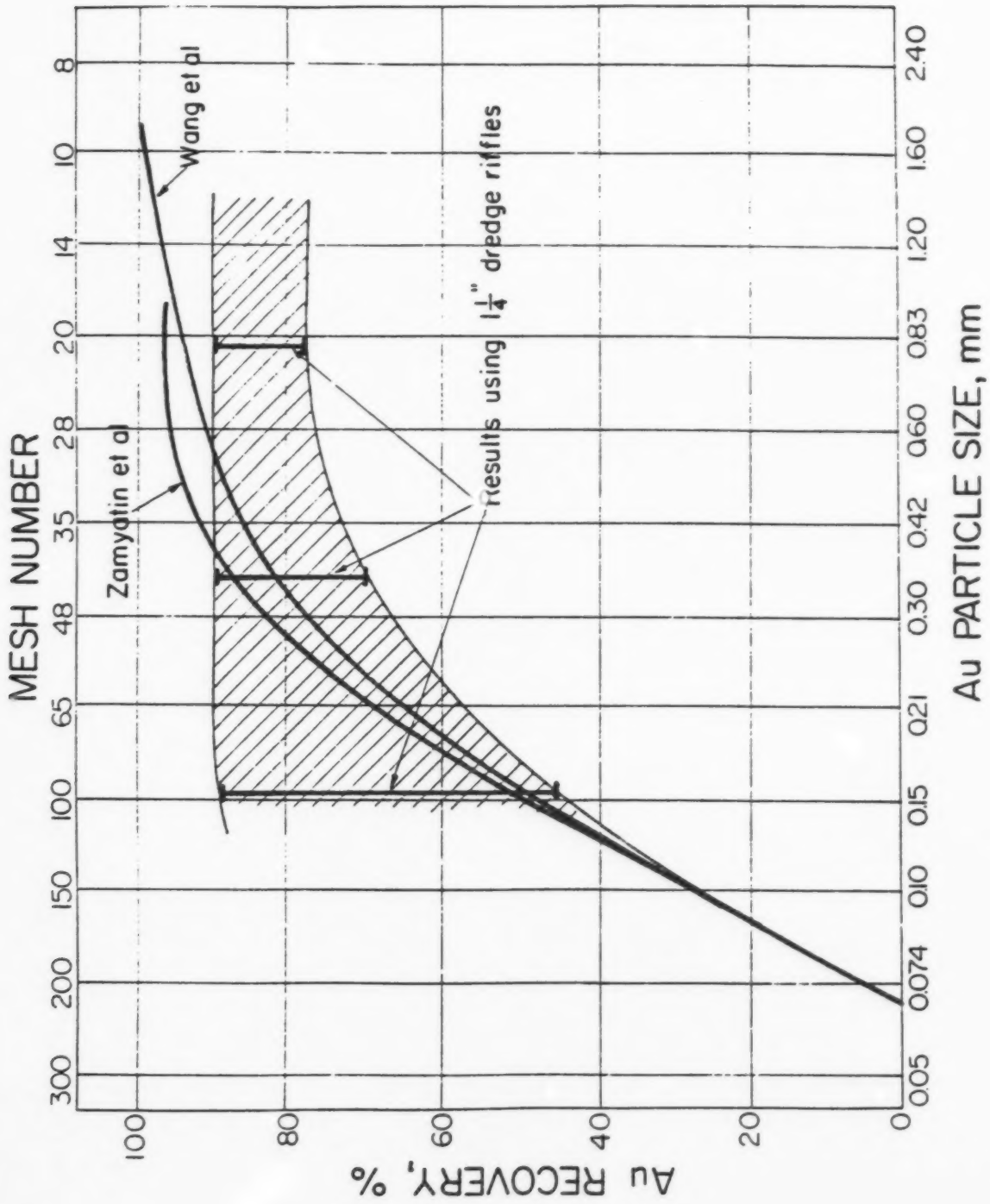


FIGURE 18 Gold Recovery Versus Particle Size Using Dredge Riffles

gold in reaching the lower portions of the concentrate trapping area where it should be less likely to be removed during scouring.

8. Recoveries in excess of 95% down to 100 mesh are possible using process water with a high suspended solids content (>10,000 ppm).
9. Low water use is beneficial to gold recovery. The best recoveries using expanded metal riffles were obtained using a water to solid ratio by weight of approximately 4:1.
10. Nomad matting and Cocoa matting are both effective at retaining gold when exposed during scouring. Nomad matting is much easier to cleanup.

7. Recommendations

1. Placer miners should use expanded metal as the sluice riffle of choice for fine gold recovery from feeds of -1" placer gravel.
2. Solid feed rates up to 700 lb/min/ft sluice width are acceptable over 1-10H expanded metal under suitable scour conditions. At this feed rate, recovery of fine gold (-65+100#) could be slightly less than 90% but overall recovery of the gold sample including the coarser gold fractions, can exceed 90%. The recommended operating procedure is to use 300-400 lb/min. of solids per foot of sluice width accompanied by approximately 200 USGPM water at a slope of $1 \frac{5}{8}$ - 2"/ft. Gold recoveries 95% of -65+100# gold should be achievable. Higher slopes, attendant with some sluice designs, would require less water flow but are more sensitive to fluctuations in solid feed rate. However lower solid feed rates, under suitable conditions, give marginally better recoveries.

3. Angle iron dredge riffles should be used somewhere in the fine gold recovery area to recover gold particles much coarser than 20 mesh and smaller than the upper feed size. Frequently referred to as a "nugget trap" when used in this manner, the dredge riffles would serve to capture gold particles too large to be retained in the relatively low profile expanded metal riffles. Except for extremely flat particles, which might be caught in the fine gold riffles, the recovery of the +10 mesh gold nuggets should be high in dredge riffles. An ideal location for such a "nugget trap" would be at the discharge end of the sluice. This location would allow the fine gold riffles to process the well sorted slurry at the sluice entrance. In this manner the maximum amount of fine gold could be recovered in the more efficient riffles prior to passing over the less efficient, more turbulent, dredge riffles. The gradient of the "nugget trap" portion of the sluice could also readily be changed to produce the appropriate scour without influencing conditions in the fine gold riffles.
4. Many valuable data could be gathered by obtaining details of cleanup results from selected, cooperative sluicing operations. Cleaning the fine gold recovery areas in sections according to distance from the feed and sizing the recovered gold would involve considerable extra time and effort but the data generated could prove beneficial to the entire placer mining industry.
5. Placer miners should investigate the effects of having short lengths of smooth, unriffled sluicebox base in their fine gold recovery sections. An example would be to have 4' of riffles at the feed end of the sluice followed by 2' of smooth base (no matting). By alternating sections of riffle and smooth base the slurry entering each riffle section would be pre-segregated so that a proportion of the high density minerals would be flowing along the base of the flow. This might counteract the tendency of the recovery probability to decrease as distance from the feed end of the sluice increases.

6. Future research on sluicing would be beneficial if directed towards investigating the recovery of gold when
 - a) using different riffle types
 - b) processing different gravel types, with or without bedrock fragments
 - c) using very fine gold (-150 mesh).

Acknowledgements

This study was jointly funded by the Government of Canada and the Yukon Territorial Government under the Economic Development Agreement for the Yukon Territory. The authors are grateful for the support from these agencies, especially that of Mr. Colin MacDonald of Mining Engineering, Department of Indian Affairs and Northern Development in Whitehorse.

The support and encouragement of the Klondike Placer Miners Association was instrumental in both the submission of the research proposal and its subsequent funding. Mr. Ron Johnson of Beron Placers provided the initial impetus followed by continued interest and support for which the authors are grateful.

The Yukon Chamber of Mines was the agency responsible for submission of the research proposal to the federal and territorial governments. The authors gratefully acknowledge the efforts of Mr. Dennis Prince and Mrs. Dorothy Howat in the administration of this project.

The gravel sample and gold used for this study were supplied by Teck Corporation from their Sulphur Creek placer mining operation. Special thanks are due to the project manager, Mr. Jerry Kline, for his cooperation and support.

Appreciation is extended to Western Canada Hydraulic Laboratory of Coquitlam, B.C. for the use of their facilities, equipment, and expertise for a nominal fee.

The authors are also grateful to Mr. Marcel Dionne of Purchasing at the University of British Columbia for his assistance during the equipment aquisition phase of this project.

Thanks also to Mary Ringwald and Debbie Craswell for typing this report.

References

1. Zamyatin, O.V. et al. The Concentration of Auriferous Sands and Conglomerates, Moscow, Nedra Press (1975).
2. Wang, Wenqian. A Study on Methods for Fine Placer Gold Processing, non-ferrous Metals (Chinese), No. 4, 6-12 (1979).
3. Wang, Wenqian and Poling, George W., Methods for Recovering Fine Placer Gold, 76, 47-56 (1983).
4. Alaska Dept. of Environmental Conservation, Placer Mining Water Treatment Technology Project, Phase 2 Report, January, 1985.

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APPENDIX

Explanation of column headings and other entries

- GOLD SIZE - size of gold particles within an upper and lower boundary expressed in Tyler Mesh Size.
- TEST SECTION - all entries on the same line as any number entry in this column are data for that test section.
- Au AVAIL - gold present in feed sample as calculated from assuming a linear loss of gold during the test program. The losses per test run were calculated by subtracting the gold recovered during tests 27 and 28 from that seeded in test 4 and dividing by 22.
- WEIGHT - weight of gold in grams of a particular size fraction retained in that sample.
- % Au AVAIL - gold recovered in sample expressed as a percentage of the calculated gold available for recovery.
- CUM % REC - cumulative recovery expressed as a percentage and based on data in column "% Au AVAIL".
- % Au REC - gold recovered in sample expressed as a percentage of the total amount of gold in that size fraction recovered.
- Au PASS SECTION - weight in grams of gold of indicated size being fed onto test section. Calculated from subtracting gold recovered in upstream test sections from "Au AVAIL".
- SECTION % REC - recovery of each test section expressed as a percentage based on values obtained from "WEIGHT" and "Au PASS SECTION". Where value of "Au PASS SECTION" was less than 1 gm the possible error in this value was considered excessive.